The Sylvia Company's Cyanide Works, Tararu, Thames, New Zealand. (Erected by Dr. A. Scheidel.)

CALIFORNIA STATE MINING BUREAU.

J. J. CRAWFORD, State Mineralogist.

BULLETIN NO. 5. San Francisco, October, 1894.

# THE CYANIDE PROCESS

ITS

## PRACTICAL APPLICATION AND ECONOMICAL RESULTS.

BY DR. A. SCHEIDEL, E.M.


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SAN FRANCISCO, October 1, 1894.

*Hon.* J. J. CRAWFORD, *State Mineralogist, San Francisco, Calif.:*

DEAR SIR: In accordance with your letter of December 8, 1893, I herewith submit my report on the cyanide process. I have endeavored to describe that process in its practical application and economical results. The information it conveys includes my own experience, and is supplemented from articles which appeared in technical periodicals; also from the records of patents granted by the Patent Offices of the United States and Great Britain, and from the Blue Books issued by the Mining Departments of the British Colonies of Australasia. I am largely indebted for special communications received from metallurgists in charge of prominent companies and important works, and from the officers of the government mining departments of the Australian Colonies. It has seemed advisable with some "improvements," and generally with the patents, to simply place them on record without any special comment.

Respectfully yours,

A. SCHEIDEL, PH.D., E.M.

# THE CYANIDE PROCESS

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# PRACTICAL APPLICATION AND ECONOMICAL RESULTS.

By A. Scheidel, Ph.D., E.M.

The "cyanide process" for extracting gold and silver from ores is based on the fact that a diluted solution of potassium cyanide dissolves these metals, forming, respectively, auro-potassic cyanide and argento-potassic cyanide, from many ores, without dissolving to any material extent the other components thereof. The process consists of treating suitable ores, when finely divided, with a weak solution of potassium cyanide, either by allowing the solution to percolate through the ore or by agitating a mixture of the ore and solution. This part of the operation being completed, the solution is separated from the solid material and the gold and silver are precipitated in metallic form. This process for the extraction of gold and silver is comparatively old in its principle, but modern in its technical application. During the last four years it has been introduced into almost every gold field, and upwards of $14,000,000 in gold and silver have been recovered by the process, which demonstrates beyond doubt that it is one of the most important additions to the wet methods of gold and silver metallurgy. The aim of this paper is to present the history of the process and to describe the ores for which it is adapted, together with their preparation and manipulation during treatment. The economical features of the cyanide process are also dwelt on at some length. The text is illustrated by plans and diagrams.

The State Mining Bureau of California was among the first in the United States to investigate the merits of the cyanide process, as set forth in a paper by Dr. W. D. Johnston, in the Xth Report of the Bureau. The process has since found extensive application, and other valuable and interesting papers have been published, but an exact account of the methods employed in all parts of the world is still wanting. This writing is undertaken at the request of Hon. J. J. Crawford, State Mineralogist of California. The facts herein recorded are obtained from the practical experience of the writer in New Zealand and the United States, and of others who have been very successful in the application of the cyanide process. I take great pleasure in expressing my sincere appreciation of and gratitude for the assistance my contributors have extended. I desire especially to acknowledge my obligation to Mr. John S. MacArthur, of Glasgow, Scotland, and to Mr. J. M. Buckland, the general manager of the African Gold Recovery Company, Lim., in Johannesburg, South African Republic. In order to allow a comparatively full description of methods and appliances, in the following pages, theoretical matter is limited to the main chemical reactions incidental to the process, and an explanation of some of the difficulties most frequently met. To facilitate the consultation of the paper, I prefix the following synopsis:

I. History of process: Solubility of gold and silver in cyanide as known to Hagen, Bagration, Elsner, Faraday; its technical application by Wright and Elkington; its metallurgical application by Rae, Simpson, Endlich and Mühlenberger, Louis Janin, Jr., Dixon, MacArthur and Forrest, Molloy, A. Janin and Merrill, W. D. Johnston.

II. Scope of process.

III. Chemistry of process.

IV. Demonstration of the process. Methods of operation:

A. The agitation process.
B. The percolation process.
  (*a*) Percolation of ores.
  (*b*) Percolation of tailings.
  (*c*) Percolation of concentrates.
C. Cyanide and cyanide solutions.
D. Treatment of the gold solutions. Recovery of the gold and silver.
  (*a*) Precipitation by zinc.
  (*b*) The Molloy process.
  (*c*) The Siemens and Halske process.
  (*d*) The Pielsticker process.
  (*e*) The Moldenhauer process.
  (*f*) The Johnston process.

V. Percentage of extraction.

VI. Working costs of process.

VII. Cost of cyanide plants.

VIII. Machinery and appliances.

IX. Laboratory work.

X. Danger in working the process.

XI. Exemplification of the process. The process in various countries:

A. Africa.
B. Australasia.
  (*a*) New Zealand.
  (*b*) Tasmania.
  (*c*) Western Australia.
  (*d*) South Australia.
  (*e*) Queensland.
  (*f*) New South Wales.
  (*g*) Victoria.
C. United States of America.
  (*a*) Utah.
  (*b*) Montana.
  (*c*) Colorado.
  (*d*) Nevada.
  (*e*) Arizona.
  (*f*) New Mexico.
  (*g*) South Dakota.
  (*h*) California.
D. Mexico, Colombia, Straits Settlements, Russia, Borneo.

XII. Summary and conclusions.

XIII. Patents:

Julio H. Rae. Improved mode of treating auriferous and argentiferous ores. U. S. patent 61,866, dated February 5, 1867.

Thomas C. Clark. Extracting precious metals from ores. U. S. patent 229,586, dated July 6, 1880.

Hiram W. Faucett. Process of treating ore. U. S. patent 236,424, dated January 11, 1881.

John F. Sanders. Composition for dissolving the coating of gold in ore. U. S. patent 244,080, dated July 12, 1881.

Jerome W. Simpson. Process of extracting gold, silver, and copper from their ores. U. S. patent 323,222, dated July 28, 1885.

John Stewart MacArthur, Robert Wardrop Forrest, M.D., and William Forrest, M.B. Improvements in obtaining gold and silver from ores and other compounds. English patent 14,174; 1887.

## XIII. Patents (*continued*):

John Stewart MacArthur, Robert Wardrop Forrest, and William Forrest Process of obtaining gold and silver from ores. U. S. patent 403,202, dated May 14, 1889.

John Stewart MacArthur. Metallurgical filter. U. S. patent 418,138, dated December 24, 1889.

John Stewart MacArthur, Robert Wardrop Forrest, and William Forrest. Process of separating gold and silver from ores. U. S. patent 418,137, dated December 24, 1889.

Edward D. Kendall. Composition of matter for the extraction of gold and silver from ores. U. S. patent, dated September 13, 1892.

Bernard Charles Molloy. Improvements in precipitating and collecting metals from solutions containing them. English patent 3,024; 1892.

John Cunninghame Montgomerie. Improvements in the extraction of gold and silver from ores or compounds containing the same, and in apparatus applicable for use in the treatment of such materials by means of solvents. English patent 12,641; 1892.

John Stewart MacArthur and Charles James Ellis. Improvements in extracting gold and silver from ores and the like. New Zealand patent-specification, June 29, 1893.

Carl Moldenhauer. Improvements in recovering gold and other precious metals from their ores. New Zealand patent-specification, August 31, 1893.

Carl Pielsticker. Improvements in the extraction of gold and silver from ores. New Zealand patent-specification, December 14, 1893.

Alexis Janin and Charles W. Merrill. Process of leaching ores with solutions of alkaline cyanides. U. S. patent 515,148, dated February 20, 1894.

William David Johnston. Method of abstracting gold and silver from their solutions in potassium cyanides. U. S. patent 522,260, dated July 3, 1894.

## XIV. List of plans, diagrams, and tables:

## XV. List of abbreviations of literature:

E. & M. J.—Engineering and Mining Journal, New York.
M. I.—Mineral Industry.
M. S. P.—Mining and Scientific Press.
Tr. A. I. M. E.—Transactions of the American Institute of Mining Engineers.
J. S. Chem. I.—Journal of Society of Chemical Industry, England.
J. fr. Chem.—Journal für practische Chemie.
J. Ch. S.—Journal Chemical Society.
Tr. Phil. Soc.—Transactions of the Philosophical Society.
M. Sc.—Moniteur Scientifique.
A. Ch. Ph.—Annales de Chimie et de Physique.
Ch. N.—Chemical News.
B. A. I. Sc.—Bulletin de l'Académie Imperiale des Sciences de St. Petersbourg.
B. S. Ch.—Bulletin de la Société Chimique de Paris.

# I. HISTORY.

The fact of gold being soluble in cyanide of potassium solution has been known for a considerable time. Hagen is reported to have mentioned it in 1806. Dr. Wright, of Birmingham, England, used gold-cyanide solution for electroplating in 1840; he made this application in consequence of his studies of Scheele's report on the solubility of gold-cyanide in a cyanide of potassium solution. J. R. & H. Elkington patented Wright's invention; they speak in their patent-specification of a boiling solution of gold or cyanide of gold in prussiate of potash. The first record in scientific literature of experiments in which metallic gold was dissolved in a cyanide of potassium solution consists in Prince Pierre Bagration's paper in the Bulletin de l'Académie Imperiale des Sciences de St. Petersbourg, 1843, t. 11, p. 136. Bagration, who alludes to Elkington's process, preserved cyanide of potassium solution in a dish, gilded on the inside. He noticed that after eight days the whole gold surface had been attacked. He experimented then with finely divided gold under the influence of the galvanic current; the latter he soon recognized as not of any benefit in the dissolving process. He precipitated the gold out of the cyanide solution by means of the electric current on a cathode of copper. Continued experiments proved the advantage of higher temperature during the dissolving process, and taught the precipitation of gold from its still warm solution by means of silver or copper plates, without the electric current. The higher temperature had, however, the disadvantage of the silver and copper being strongly attacked by the cyanide solution during the precipitation process. Bagration extended his experiments to solutions of ferrocyanide, which he found to act like cyanide, but in a much less degree. He further studied the solubility of gold in the form of plates, in cyanide, and found it to be dissolved in such form at a considerable rate at a temperature of 30° to 40° C. He noticed the influence of the air on the reaction. Bagration believes that hydrocyanic acid in a state of generation is a gold solvent, and he concludes his paper with the remark that in the future, cyanide of potassium must be enumerated among the solvents of gold. L. Elsner published in J. fr. Chem., 1844, p. 441, his observations on the reactions of "reguline metals" in an aqueous solution of cyanide. He found that gold and silver were dissolved in potassium cyanide without decomposition of water. "The dissolution of the metals is, however, the consequence of the action of oxygen, which, absorbed from the air, decomposes part of the cyanide." His reaction has been expressed by others in the following equation:

$$\underset{\text{(Gold.)}}{2\,Au} + \underset{\text{(Cyanide of potassium.)}}{4\,KCy} + \underset{\text{(Oxygen.)}}{O} + \underset{\text{(Water.)}}{H_2O} = \underset{\text{(Auro-potassic cyanide.)}}{2\,AuKCy_2} + \underset{\text{(Potassic hydrate.)}}{2\,KOH}$$

It is generally called Elsner's equation. Some years after, Faraday made use of the solubility of gold in cyanide solution for reducing the thickness of gold films (Exp. relations of gold and other metals to light, Tr. Phil. Soc., 1857, p. 147). The basis of the most modern process for the extraction of gold was thus provided. It took many years, however, before the enumerated facts were made use of for the extraction of gold from ores. In 1867, Julio H. Rae took out United States patent No. 61,866, dated February 5th, for an "improved method of

treating auriferous and argentiferous ores" with a current of electricity in connection with suitable liquids—such, for instance, as cyanide of potassium. Rae's process is an agitation process; he proposed to "expose the auriferous or argentiferous rock to the combined action of a current of electricity and of suitable solvents, and to separate the gold or silver from the rocks containing the same by the action or aid of electricity." The principle of Rae's process, as stated by him, distinguishes his method from the modern cyanide process. His method does not appear to have advanced beyond the laboratory stage or to have found extensive and successful practical application, and it sank into oblivion. Since then, cyanide of potassium in connection with gold and silver metallurgy has repeatedly been made a patent claim; in many cases, however, the application recommended is in its principle different from the application which characterizes the modern cyanide process. Thomas C. Clark, of Oakland, Cal. (United States patent No. 229,586, July 6, 1880), roasted his ore to a red heat, and placed it in that condition in a cold bath composed of a solution of salt, prussiate of potash, and caustic soda. H. W. Faucett, of St. Louis, Mo. (United States patent No. 236,424, January 11, 1881), subjects hot crushed ores to the action of disintegrating chemicals, cyanide of sodium among others, in solution under pressure, the pressure being effected by the steam generated by the contact of the hot ores with the chemical solution in a closed vessel. This treatment, like that proposed by Clark, was intended as preliminary to amalgamation. John F. Sanders, of Ogden, obtained United States patent No. 244,080, dated July 12, 1881, for "composition for dissolving the coating of gold in ore." This composition is made of cyanide of potassium and glacial phosphoric acid. He stated that by using this mixture he could dissolve "the impure coatings of gold, leaving the gold free and exposed, and permitting it to be amalgamated." It is evident, therefore, that these processes bear no similarity or relation to the modern cyanide process. For a considerable time, cyanide of potassium has been used in the gold fields of California and Australasia for removing film-coating from gold in ores; its application in the pan-amalgamation process may have been a source of loss of gold.

The application of a cyanide of potassium solution for the extraction of gold and silver direct from their ores, which application had been neglected since Rae, was taken up again by Jerome W. Simpson, of Newark, N. J., who obtained United States patent No. 323,222, dated July 28, 1885, for a process of extracting gold, silver, and copper from their ores. Simpson reduced his ore to a powder and agitated it with a solution of certain salts, which combine chemically with the metal in the ore and form therewith a soluble salt. The salt solution was composed of one pound of cyanide of potassium, one ounce of carbonate of ammonia, one half ounce of chloride of sodium, and sixteen quarts of water. This solution is described as particularly adapted to ores containing gold, silver, and copper in the form of sulphurets. In the absence of silver no chloride of sodium is used; for ores rich in silver a proportionately larger quantity of chloride of sodium is employed. The metals dissolved in the salt solution were precipitated by means of zinc, suspended therein in form of pieces or plates. Simpson was aware that cyanide of potassium, in connection with an electric current, had been used for dissolving metal, and also that zinc had been employed as a precipitant. What he claims as new

is: (1) The process of separating gold and silver from their ores, which consists in subjecting the ore to the action of the solution of cyanide of potassium and carbonate of ammonia, and subsequently precipitating the dissolved metal by means of zinc. (2) The process of separating metals from their ores, to wit, "subjecting the ore to the action of a solution of cyanide of potassium, carbonate of ammonia, and chloride of sodium, and subsequently precipitating the dissolved metals."

My own experiments have proved that the addition of sodium chloride is of no benefit for the extraction of silver. The addition of ammonium carbonate is not beneficial to the extraction of either gold or silver, except under certain conditions, when it may be substituted advantageously by an alkali or an alkaline earth; in the presence of base metals it is of disadvantage. Simpson's patent description appears to indicate that he had not discovered the most important property of dilute cyanide solutions, namely, that of dissolving, without the addition of other chemicals, the noble (in preference to the base) metals. His patent-claim consists eminently in adding to the cyanide solution the chemicals mentioned above. His process, like that of Rae, is an agitation process. The zinc for precipitating the bullion he used in the form of plates or pieces, There is no record in the technical literature in reference to the application of Simpson's process before the issue of the MacArthur and Forrest patents in 1889. After Rae and Simpson, others have made experiments with cyanide solutions for the purpose of gold and silver extraction from ores. F. M. Endlich and N. H. Mühlenberger are reported to have filed a caveat in 1885, without, however, securing a patent, the former having apparently become doubtful as to the applicability of cyanide as an economical process (E. & M. J., 1891, p. 86). Louis Janin, Jr., made interesting experiments in the same direction in Park City, Utah, of which he published the results in 1888 (E. & M. J., 1888, p. 548). These experiments refer chiefly to silver extraction, but he mentions as well the results on gold ores; his results appear to have been encouraging, and led to his filing a caveat on May 1, 1886, but this was not pushed on to the taking out of a patent.

In the southern hemisphere, W. A. Dixon has made experiments with cyanide on Australian ores as early as 1887. He recorded his results, which are at least of historical interest, in a paper read before the Royal Society of New South Wales. Dixon describes therein the experiments made by him at the instigation of the Government Inspector of Mines, who suggested that the extraction of gold from complex minerals was a subject worthy of investigation. Dixon tried on such ores amalgamation and a number of solvents. He found "the aurocyanides of the alkaline metals of sufficient stability to render their use possible for the extraction of the gold." He mentions Bagration's and Elsner's publications and alludes to Rae's patent, of which, however, he possessed no particulars. Dixon feared that "the high price of cyanide, its instability when exposed to the air, and its extremely poisonous qualities," would prove such obstacles as to preclude its use for metallurgical purposes. He found the reaction between gold and cyanide slow if "the gold was at all dense"; in presence of alkaline oxidizing agents, however, he found the dissolving process sufficiently rapid. Dixon experimented also with the ferrocyanide of potassium. His results generally, did not, as far as known, lead to the metallurgical application of cyanide as a gold and silver solvent.

I have thus described the history of cyanide of potassium as a gold dissolving agent from the early laboratory experiments up to its metallurgical application for ore extraction; this latter, however, did not gain any practical importance until John S. MacArthur and W. Forrest, of Glasgow, Scotland, took out their patents for the use of cyanide as a gold and silver solvent from ores, and gave thereby the cyanide process a start all over the world. Their patents mark an epoch in gold metallurgy. The results of the application of cyanide, as suggested by them, have been very satisfactory; the $14,000,000 of bullion produced by it during the five short years of its working represents what, by the ordinary methods, would have been irrecoverably lost; hence its value and importance from the standpoint of metallurgy and political economy. The experiments of MacArthur and Forrest with gold dissolving reagents occupied some years before their English cyanide patents were applied for (J. S. MacArthur, J. S. Chem. I., March 31, 1890, No. 3, vol. 9). They drew out a list of possible solvents having a stronger affinity for gold than for sulphides, which included the cyanides, and which they found to solve the problem. Their experiments, conducted first on a small scale and with ores of many kinds and of different sources, were so satisfactory that they gradually worked on a larger scale, and their results formed the basis for the introduction of the cyanide process into most gold-producing fields. Their English patent was applied for October 19, 1887. Since then they applied for and obtained patents in many gold-producing countries. Their United States patents are dated as follows: 403,202, May 14, 1889; 418,137, December 24, 1889; 418,138, December 24, 1889. Their invention is described "as having principally for its object the obtaining of gold from ores, but it is also applicable for obtaining silver from ores containing it whether with or without gold, and it comprises an improved process, which, while applicable to auriferous and argentiferous ores generally, is advantageously and economically effective with refractory ores, or ores from which gold and silver have not been satisfactorily or profitably obtainable by the amalgamating or other processes hitherto employed; such as ores containing sulphides, arsenides, tellurides, and compounds of base metals generally, and ores from which the gold has not been easily or completely separable on account of its existing in the ores in a state of extremely fine division." The patentees describe their invention (I am following United States patent 403,202) as consisting in subjecting the ores to the action of a solution containing a small quantity of cyanide, without any other chemically active agent. In dealing with ores containing per ton twenty ounces or less of gold or silver, or gold and silver, they find it most advantageous to use a quantity of cyanide, the cyanogen of which is equal in weight to from one to four parts for every thousand parts of the ore dissolved in a quantity of water of about half the weight of the ore; they generally use a solution containing two parts of cyanogen for every thousand parts of the ore. In the case of richer ores, while increasing the quantity of cyanide to suit the greater quantity of gold or silver, they also increase the quantity of water so as to keep the solution dilute; in other words, the cyanide solution should contain from two to eight parts, by weight, of cyanogen to one thousand parts of water, and the quantity of the solution used should be determined by the richness of the ore. The patentees state: "By treating the ores with the dilute and simple solution of a cyanide, the gold or silver is, or the gold and silver are,

obtained in solution, while any base metals in the ores are left undissolved except to a practically inappreciable extent; whereas, when the cyanide is used in combination with an electric current, or in conjunction with another chemically active agent, such as carbonate of ammonia, or chloride of sodium, or phosphoric acid, or when the solution contains too much cyanide, not only is there a greater expenditure of chemicals in the first instance, but the base metals are dissolved to a large extent along with the gold or silver, and their subsequent separation involves extra expense, which is saved by their process." Later on MacArthur and Forrest obtained patents covering the use of zinc in a fine state of division for the purpose of precipitating gold and silver from cyanide, chloride, bromide, thiosulphide, sulphate, or other similar solutions; they further protected the use of an alkali or alkaline earth for neutralizing ores preparatory to subjecting the same to the action of cyanogen or of a cyanide. The MacArthur-Forrest patent-claims consist, therefore, in three points: (1) The application of diluted solutions of cyanide (not exceeding eight parts of cyanogen to one thousand parts of water); (2) the use of zinc in a fine state of division; (3) the preparatory treatment of the ore, which has become partially oxidized by exposure to the weather, with an alkali or alkaline earth, for the purpose of neutralizing the salts of iron or other objectionable ingredients formed by partial oxidation.

It is not the purpose of this paper, which is intended to describe the historical development of the cyanide process and its present forms of application, to enter into a judicial discussion of patent-claims and patent-rights. It is the duty of the historian to date the cyanide process as a commercial success from 1890, when it was introduced as "the MacArthur-Forrest process" on the Witwatersrand gold fields, in the South African Republic. Its success as a metallurgical experiment may be dated from the tests made on a large scale with ore from the New Zealand Crown Mine in June and July, 1888. The practicability of the cyanide process once established, others endeavored to introduce improvements in its application, which they protected by letters patent. A patent which once promised to become of practical importance is that of B. C. Molloy, of Johannesburg, whose "improvement" consists in the abolition of zinc as a precipitant of gold, and in the revivification of the cyanide of potassium in the solution. In this process the ore is treated with cyanide of potassium as usual; the resulting liquors are passed through a "patent Molloy separator," which consists of an amalgamator, the mercury of which is constantly being charged electrolytically with potassium. The potassium on coming into contact with the water of the solution, decomposes it with the evolution of hydrogen and the formation of the oxide of the alkaline metal. The nascent hydrogen decomposes the solution of the cyanide of gold, and sets the gold free, which is precipitated upon and collected by the mercury; the metal of the alkaline oxide reacts upon the cyanogen compound, and so reproduces the cyanide of potassium. The original solution, thus regenerated, is then ready for use again. (In reference to further details see under precipitation of gold and silver, p. 38.)

Among other cyanide patent-specifications may be mentioned the following: John C. Montgomerie, of Scotland, obtained English patent No. 12,641, 1892, "for improvements in the extraction of gold and silver from ores and in apparatus applicable for use in the treatment of such

by means of solvents." His process is the well-known agitation process of finely divided ore with cyanide solution and the addition of an alkaline oxide "for the purpose of economizing the solvent and expediting its action." The patent-specification does not contain any claims which might be termed either an invention or an improvement, either chemically or mechanically. One of the latest additions to the cyanide patent literature is United States patent 515,148, dated February 20, 1894, of Alexis Janin and Charles W. Merrill, for a process of leaching ores with solutions of alkaline cyanides. (For patent-specification, see Appendix.) They claim as new "the art of leaching ores with solution of alkaline cyanides, which consists in first leaching the ore with such solutions, then adding to the solution an agent which will precipitate the silver present as a sulphide, and then precipitating the gold in the solution with metallic zinc." The practical advantages of this complication will have to be proved.

A patent description of interest, although not a "cyanide process" strictly speaking, is that of E. D. Kendall, of Brooklyn, N. Y., dated September 13, 1892, who claims the use of potassium ferrocyanide combined with cyanide of potassium, for extracting gold and silver from ores, etc., as his invention. (For patent-specification, see Appendix.)

A further addition to the patent literature is the specification of MacArthur and Ellis, who propose to increase the efficiency and economy of the process in cases in which from the nature of the ores treated or other circumstances, soluble sulphides are formed, which retard and objectionally affect the action of the cyanide on the precious metals by adding to the ore or the cyanide solution suitable salts or compounds of metals which will form with the sulphur of the soluble sulphides an insoluble or inert sulphide. For this purpose preference is being given to the metallic salts or compounds in the following order: Salts or compounds of lead—such as plumbates, carbonates, acetate or sulphate of lead—sulphate or chloride of manganese, zincates, oxides, or chloride of mercury, ferric hydrate or oxide. The proportion to be used is easily to be ascertained by trials of a few samples in each case. (See patent-specification of John Stewart MacArthur and Charles T. Ellis, in Appendix.)

C. Moldenhauer proposes to render the cyanide process more expeditious and considerably cheaper by, firstly, adding to the cyanide solution an artificial oxidizing agent, by preference ferricyanide of potassium in alkaline solution, and, secondly, in precipitating the extracted precious metal out of its cyanide solution by means of aluminium, or alloys, or amalgam thereof. (See patent-specification in Appendix.)

C. M. Pielsticker reverts to the application of the electric current, in conjunction with the cyanide solution. He proposes to continuously circulate the solvent, to continuously precipitate the dissolved precious metals by electrolysis, and continuously regenerate thereby the reagent. (See patent specification in Appendix.)

The latest patent in connection with cyanide treatment of ores is that of Dr. W. D. Johnston, "for abstracting gold and silver from their cyanide solutions by means of pulverized carbon" (for further details, see page 40).

To make this report as complete as circumstances permit, I append the specifications of the patents which have been mentioned in the body

of this paper, that the mining public may know the exact wording of descriptions and claims.

Such is the history of the cyanide process, rapidly sketched by tracing its development through the phases of its evolution and the intricacies of its patent literature. The modern cyanide process consists in the treatment of ores by means of dilute cyanide of potassium solutions, as a rule without the addition of other chemical substances, and in the subsequent precipitation of the gold and silver from the solution by means of zinc in form of shavings. It is commonly known as the MacArthur-Forrest process. I now propose to enter into a description of the process itself. I embody in it the information given me by Mr. John S. MacArthur, of Glasgow, Scotland.

## II. SCOPE OF PROCESS.

The process can be advantageously applied to many gold ores and many silver ores, and is often suitable for ores which are generally considered as rebellious or refractory. The word "ore" is here meant to include ores, tailings, concentrates, and all similar products from ore. The term "refractory" is used to signify any ore which cannot be satisfactorily amalgamated. The refractory character of such ores can be caused by the presence of base metals in combination with sulphur or arsenic, or otherwise by their physical structure, which prevents the gold from coming in contact with the mercury during the amalgamation process. To the latter class belong the ores in which the gold is "coated" with substances which prevent metallic contact ("rusty gold"). (An excellent instance of such coated ore is that found in the Mount Morgan Mine, in Queensland, where the finely divided gold is coated with a film of what has been termed hydrous peroxide of iron, which makes the gold absolutely refractory to amalgamation.) To the same class of refractory ores belong those in which the gold is so finely divided that the film of air surrounding the auriferous particles prevents amalgamation even under the most favorable conditions. The base metals which most frequently accompany refractory ores are iron, zinc, lead, copper, and antimony—usually as sulphides, sometimes as arsenides. When ores containing gold, silver, copper, zinc, iron, etc., are treated with solutions of cyanide of potassium, these metals are dissolved more or less, forming soluble cyanides. The solvent action on the base metals can be reduced to a minimum by reducing the strength of the solutions, the readily soluble gold and silver being easily dissolved out with only traces of copper, zinc, etc. The action of these weak cyanide solutions on iron, lead, arsenic, etc., is practically *nil*, and the solvent action on copper or zinc depends much upon the state of chemical combination in which they exist.

The cyanide process is adapted to treat most of such refractory ores as are described above. The principal exceptions are the ores which contain hydrated copper oxides and copper carbonates, and those which contain an appreciable quantity of antimony. "When copper compounds exist in a state physically hard, the cyanide solution does not readily act on them; but when the copper compounds are soft, porous, and spongy, the action of the cyanide is so decided as to interfere materially with its action on gold" (MacArthur). In reference to copper

sulphide I found it no impediment to the process; carbonate of copper, however, was so readily attacked by cyanide that its presence proved absolutely prohibitive to the extraction of silver and interfered seriously with the extraction of gold. This most refractory ore, that I am speaking of, came from old workings in the Sylvia Mine, Tararu, New Zealand, where part of the ledge containing a large percentage of copper pyrites had been exposed for many years to the influence of moisture and the atmosphere; the resulting carbonate was hard, but notwithstanding this its reaction on cyanide solutions was very marked. One and a fourth ounce of such copper ore, finely divided and shaken for less than fifteen minutes with a 2.73 per cent cyanide of potassium solution, reduced the strength of the solution to 0.05 per cent of cyanide. The treatment of the ore in question proved that the affinity of cyanide to gold is at least equal to that of cyanide to copper, and very much greater than to silver, as, notwithstanding the rapid consumption of cyanide by the copper compound, upwards of 70 per cent of the gold assay-value was extracted by cyanide solution of the usual strength, whereas at the same time absolutely no silver had gone into solution. A preliminary treatment of such ore by sulphuric acid had a beneficial effect on the consumption of cyanide and thereby on the extraction of silver.

"In the case of antimonial ores, there is little or no interaction between the antimony and the cyanide, consequently the latter is not taken up; but as gold seems to be very firmly held by antimony, and as the compound is very impervious, the cyanide is unable to penetrate the mass, and to dissolve and separate the precious from the base metals. In the case of both copper and antimony the cyanide solution will act, but in the case of copper, if there is much present and acted upon, the consumption of cyanide is so great that the operation is not profitable, and in the case of the antimonial ores, though the cyanide will act with fine grinding and long contact, the expense involved often overbalances the value of the gold contents" (MacArthur). The physical state in which obnoxious compounds are found, is of the greatest importance. Hard-surfaced crystals are, even if finely divided, naturally less acted upon by cyanide than soft, spongy masses of the same size. For technical purposes, cyanide treatment of any ore will be called unsuccessful if the large consumption of cyanide precludes a commercial success, although finally a satisfactory extraction in percentage may be achieved.

## III. THE CHEMISTRY OF THE PROCESS.

The chemical reaction on which the cyanide process of gold extraction rests is that of the formation of the double cyanide of gold and potassium:

$$\underset{\text{(Gold.)}}{2\,Au} + \underset{\text{(Cyanide of potassium.)}}{4\,KCy} + \underset{\text{(Oxygen.)}}{O} + \underset{\text{(Water.)}}{H_2O} = \underset{\text{(Auro-potassic cyanide.)}}{2\,AuKCy_2} + \underset{\text{(Potassic hydrate.)}}{2\,KOH}$$

That of silver extraction produces the double cyanide of silver and potassium:

$$\underset{\text{(Silver.)}}{2\,Ag} + \underset{\text{(Cyanide of potassium.)}}{4\,KCy} + \underset{\text{(Oxygen.)}}{O} + \underset{\text{(Water.)}}{H_2O} = \underset{\text{(Argento-potassic cyanide)}}{2\,AgKCy_2} + \underset{\text{(Potassic hydrate.)}}{2\,KOH}$$

Silver in the metallic state is, however, rarely met with in ores which are subjected to cyanide treatment. The part taken by oxygen in these reactions, first noticed by Prince Bagration and later confirmed by Elsner, has of late been disputed, but again confirmed by McLaurin, who published his experiments (J. Ch. S., 1893, May, p. 724) in reference to the question, and came to the following conclusions: (1) That oxygen is necessary for the solution of gold in cyanide of potassium, and that it combines with the potassium of the potassium cyanide in the proportions required by Elsner's equation; (2) That the rate of solution of gold in a solution of potassium cyanide passes through a maximum in passing from dilute to concentrated solution, and this remarkable variation is capable of explanation by the fact that the solubility of oxygen in a cyanide solution decreases with the concentration. The double compounds of cyanide of potassium and gold and silver, respectively, have been described in the Annales de Chimie et de Physique, 53, p. 462, 1858, and in Bull. de la Société Chimique de Paris, 29, 1878, p. 460. Both compounds are easily soluble in water.

The cyanide process, as illustrated by the before-mentioned equations, appears very simple indeed. Its adoption in many places has been very rapid, and its success, particularly on the tailings of the Johannesburg mills, has been great. The practical working and technically successful carrying out of cyanide treatment of any ore, even under the most favorable circumstances, is beset with complications, which require a careful study of all the circumstances connected with the case. All operations offer occasions for loss and opportunities for improvement. The reaction between cyanide and the metals, so simple in theory, is in practice more or less complicated by the reaction of other ore compounds on the cyanide and by other causes which it will be useful to investigate. That such reactions take place is put in strong evidence by the amount of cyanide consumed in treating ores, which is always considerably larger than the quantity theoretically necessary to dissolve the gold. In accordance with Elsner's equation, 10 parts of cyanide should dissolve 15.12 parts of gold. In the works at Johannesburg, however, in treating free-milling ore, 40 parts of cyanide are required to dissolve one part of gold; that is to say, 40 parts of cyanide are consumed for each part of gold obtained. The main reason for this fact must be looked for in secondary reactions, which as yet have only been partly studied. The great loss of cyanide takes place during the extraction process, and particularly during the first part of it, as proved by the rapid diminution in the strength of the solution. The loss of cyanide in the zinc boxes has often been exaggerated (see page 34). A loss of cyanide occurs by absorption in vats and tanks, which is given as high as one pound per ton of ore in Johannesburg (Butters and Clennell). Some loss will always result from the action of carbonic acid gas, which is always present in the atmosphere, and displaces cyanogen from the alkali, setting prussic acid free, which escapes into the air; if caustic alkali is present the freed prussic acid will be neutralized. The extent of loss by hydrolysis requires further investigation. The presence of free sulphuric acid or other products of more or less advanced decomposition of pyritic matter will naturally considerably interfere with the simple reaction by increasing the consumption of cyanide, and may, under the most unfavorable circumstances, completely prevent successful treatment. "In many cases tailings which

have been exposed to the weather contain oxidized compounds, such as sulphate of iron, and similar sulphates of alumina and magnesia, formed by the action of the metallic sulphates on the earthy constituents of the ore." When this is the case it is advisable to give such tailings one or more preliminary water-washings, because the cyanide is partly absorbed and partly decomposed by these substances, as seen in the following equations:

$$FeSO_4 + 2\ KCy = FeCy_2 + K_2SO_4$$
$$Fe_23(SO_4) + 6\ KCy + 3\ H_2O = Fe_2O_3 + 3(K_2SO_4) + 6\ HCy$$

From these equations it will be seen that the ferrous oxide combines with cyanogen, and that the sulphuric acid, forming the second constituent of the ferric salt, liberates hydrocyanic acid, which being volatile is not available, and moreover constitutes a loss and a danger. The action of the sulphates of alumina and magnesia has not been generally and sufficiently recognized. These salts act practically as if they were sulphuric acid: hydrocyanic acid is liberated and alumina or magnesia, as the case may be, precipitated, as shown in the following equations:

$$Al_23(SO_4) + 6\ KCy + 3\ H_2O = Al_2O_3 + 3\ (K_2SO_4) + 6\ HCy$$
$$MgSO_4 + 2\ KCy + H_2O = MgO + K_2SO_4 + 2\ HCy$$

The remedy for these troubles is, as before stated, water-washing, in some cases followed by a lime or soda treatment. Reference only has been made to ferrous sulphate as a soluble salt, but it has been found that the basic ferrous salts, which exist to a greater or less extent in "weathered" tailings, are insoluble in water, and yet act detrimentally on cyanide. In any case it is difficult to wash out the last traces of any soluble substance, and it is wise to economize cyanide by an alkaline treatment. "While ferrous salts, soluble or insoluble, exist in the tailings, the lime or soda combines with the acid and deposits the ferrous oxide or hydrate in the tailings. The ferrous oxide would still absorb cyanogen if a cyanide solution were present, but if the air has free access before a cyanide solution is applied, the ferrous oxide is oxidized to ferric oxide, which does not combine directly with cyanogen. It will thus be seen that where salts of iron have to be dealt with it is advisable to make the alkaline treatment preliminary to permit of the necessary oxidation; but where sulphates of alkaline earths only are in question, the requisite lime or alkali may be added along with the cyanide solution. Where soluble iron salts are present to any extent, the washing should be very thorough, and the solution should be run off from the vat through a separate pipe which has no connection with any of the cyanide pipes" (or better, the washing should take place in a special vat; see page 49). "This matter of salts formed by oxidation arises chiefly in the case of tailings, but it may also happen with concentrates and ores, in which case they are treated as tailings" (MacArthur).

Butters and Clennell advance the following equations of possible reactions accompanying the action of cyanide on pyrites. They illustrate first the influence of oxygen on pyrites:

$$FeS_2 + H_2O + 7\,O = FeSO_4 + H_2SO_4$$

$$2\ FeSO_4 + O = Fe_2O_3,\ 2\ SO_3\ \text{(Wittstein)}$$

$$10\ FeSO_4 + 5\,O = \underset{\text{(Basic sulphate insoluble.)}}{2\ Fe_2O_3,\ 2\ SO_3} + \underset{\text{(Ferric sulphate soluble.)}}{3\ Fe_2(SO_4)_3}\ \text{(Berzelius)}$$

They describe, then, the reaction of cyanide on such products:

$$FeSO_4 + 2\ KCy = FeCy_2 + K_2SO_4$$
$$FeCy_2 + 4\ KCy = K_4FeCy_6$$

Ultimately giving rise to

$$3\ K_4FeCy_6 + 6\ FeSO_4 + 3\ O = Fe_2O_3 + 6\ K_2SO_4 + Fe_7Cy_{18}$$

Ferric salts and cyanide give:

$$Fe_2(SO_4)_3 + 6\ KCy = Fe_2Cy_6 + 3\ K_2SO_4$$

and,

$$Fe_2Cy_6 + 6\ H_2O = Fe_2(OH)_6 + 6\ HCy$$
$$Fe_2(SO_4)_3 + 6\ KCy + 6\ H_2O = Fe_2(OH)_6 + 6\ HCy + 3\ K_2SO_4$$

A mixture of ferrous and ferric sulphates on addition of cyanide will form Prussian blue when the ferric salt is in excess:

$$18\ KCy + 3\ FeSO_4 + 2\ Fe_2(SO_4)_3 = 9\ K_2SO_4 + Fe_4(FeCy_6)_3$$

and Turnbull's blue when ferrous salt is in excess:

$$12\ KCy + 3\ FeSO_4 + Fe_2(SO_4)_3 = 6\ K_2SO_4 + Fe_3(FeCy_6)_2$$

The reactions between the various iron and cyanogen compounds are very complicated, and a number of possible reactions have been illustrated by equations by various writers, the discussion of which here would take up too much space.

In cases where such conditions exist, a preliminary washing with water alone, or with solutions of carbonates or hydroxide of sodium or lime, as described, may be not only useful but imperative. A great surplus of alkali should be avoided, on account of its action on the zinc in the precipitation boxes. The loss of zinc will be larger the greater the alkalinity of the solution; besides this, it is apt to form a sulphide of sodium or potassium with the sulphur of ores, which interferes with the extraction of the silver. Further careful scientific researches in reference to secondary reactions, which accompany the cyanide process, will probably lead to technically important results. The chemistry of precipitating the metals from cyanide solution will be discussed in connection with the description of the various methods employed for that purpose.

## IV. DEMONSTRATION OF THE PROCESS—METHODS OF OPERATION.

The cyanide process is worked either by agitating the ore with the solution ("the agitation process"), or by allowing the solution to pass through the ore ("the percolation process").

### A. The Agitation Process.

When cyanide treatment of ores was first attempted, it was done by agitating the material under treatment with cyanide solution; Rae's cyanide process of 1867 and Simpson's process of 1885 were agitation processes. Generally speaking, agitation, as compared with percolation, expedites and in instances increases extraction, but it requires motive power, which is a source of expense. Wherever large quantities of ore are being treated it has been abandoned in favor of the percolation process. It is useful, however, in many instances where the ores are hard and dense, and of a sufficient high value to pay for the necessary motive power and permit a convenient method of filtration; it is applied where the quantities are limited and is mostly used for treating concentrates, or such ores as make the treatment of limited parcels by themselves desirable. The importance of the cyanide agitation process has not been so fully recognized as, in some instances, it deserves. It is natural that if percolation gives as cheaply the same results it will be preferred, but sometimes the agitation system has the advantage of giving quicker, higher, and cheaper returns. Some ores, particularly ores containing tellurides and sulphide of silver, give better results by agitation than by percolation. The agitation process, in its present form, is not well adapted to handling very large quantities of ore without a considerable outlay of machinery. Technical improvements of the system, which in suitable cases may make the whole process almost a continuous one, may be expected. The chief appliances for the agitation process are the agitator and the filter. Although any vat fitted with revolving arms and barrels, similar to those employed in chlorination, may be used successfully for agitation, still an agitator which permits a charge and discharge quickly and safely, which has the least wear and tear, does absorb neither gold nor cyanide, and is cheap in its first cost, corresponds best with all requirements. I have been using wooden barrels, wooden vertical agitators, iron pans, and steel cylindrical agitators, and have found the latter construction best suited to the purpose and satisfying all the above conditions. (For description of such an agitator, see Utica cyanide plant, page 89.)

For the purpose of extraction, the ore and the cyanide solution are agitated for a time, varying in accordance with the character of the ore, generally ranging from six to twelve hours. I have extracted from complex ores, in some instances, upwards of 90 per cent of the assay-value in less than two hours, and in other instances I have found it necessary to continue the operation for twenty-four hours. No general rule can be given; each case has to be investigated and the *modus operandi* to be selected according to circumstances. (See table showing rate of extraction in relation to time of agitation, attached to the description of the Utica plant, page 94.) The strength of the cyanide solution and the

volume required depend entirely on the character of the ore; as a rule, solutions for agitation should be stronger than those for percolation. Here, like in other matters in connection with cyanide treatment, experimental investigation has to advise on best conditions. (See chapter on laboratory work, page 44.) In using barrels as agitators, ore and solution will be charged before the barrel is revolved; if vertical vessels are used, the solution will be charged first, then the stirrer will be set in motion, and the ore added by degrees.

When the extraction is completed, the mass in the agitator is discharged, and the cyanide solution, now containing the gold, is separated from the solid material—*i. e.*, the residues—by any method which local conditions and the character of the ore suggest. Apparatus of different principles have been used for this purpose. Filter presses of various constructions, vacuum filters, and centrifugal machines have been employed. Concentrates, coarse and slimy, can be successfully treated by means of my vacuum filter (see description of Sylvia and Utica plants, p. 79 and p. 89), which permits a quick filtration and a perfect and speedy washing of the residues with a minimum of liquid. In some instances I made successful use of centrifugal force for separating the gold solution and washing the residues. For washing, weak cyanide solutions from previous operations are used, and finally a water-wash is given. The residues are then discharged. The gold solutions should, for practical reasons, be kept separate according to their strength in gold and cyanide; they pass through such appliances as are used for precipitating the gold and silver, after which the "liquors" are collected in sumps for use on subsequent charges of ore. In well-appointed works no cyanide solution is allowed to run to waste, as the same amount of liquid remains constantly in circulation.

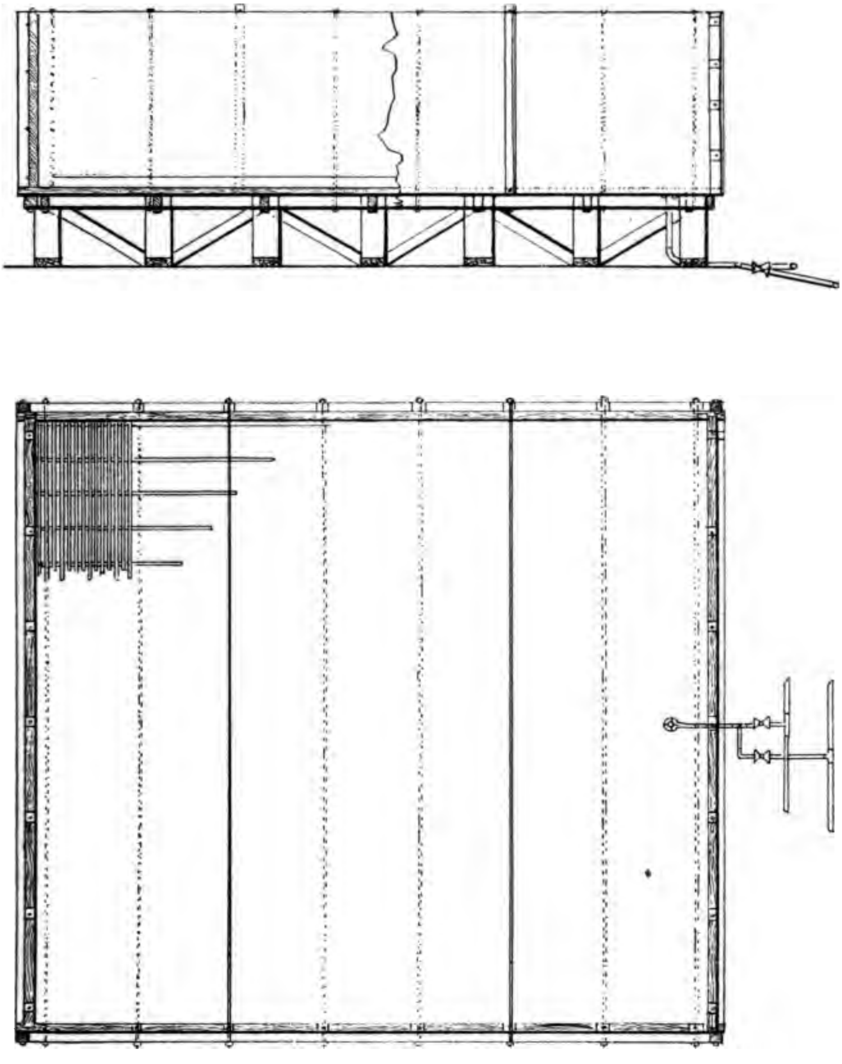
The author took out early in 1893 a caveat in New Zealand for a centrifugal apparatus, agitator and separator combined, for the treatment of slimy ores by agitation with cyanide, and subsequent separation of the gold solution by centrifugal force in the same apparatus. Experiments have of late been made in the Thames School of Mines, New Zealand, with the treatment of slimy ores by the agitation process in an apparatus which is described as follows: The appliances used in the operation consist of a shallow circular vat, a vacuum cylinder, and an air pump. The vat is provided with four revolving arms, to which soft rubber brushes are fixed. The bottom of the vat is fitted with a false bottom, constructed of a wooden grating covered with wool packing. The operation is conducted as follows: The leaching solution, made up to the required strength, is first conducted into the vat. The revolving arms are then set in motion, and the dry pulp or slimes introduced. The agitation is continued for six hours, or until the extraction is complete. A stopcock in a pipe connecting the false bottom of the leaching vat and vacuum cylinder, is then opened and the air pump started. The effect is immediate. At once the clear solution begins to drain over into the cylinder, the revolution of the arms preventing the slimes from settling and choking up the filter cloth. When the slimes have been drained down to a thick paste, the first wash water is added, the pump again started, and the slimes drained as before. The water-washings are carried on in the same way, and when completed a plug or door is opened and the leached slimes are sluiced out. The whole operation of leaching takes from eighteen to twenty-four hours. The technical and economical practi-

cability of this method of treating slimy ores appears doubtful and will have to be proved.

## B. Percolation Process.

Percolation is the method generally in use. It is being worked in the United States, in the British Colonies of Australasia, and on a very extensive scale in the South African gold fields, and therefore merits a full description. Percolation consists in soaking cyanide solution through ore. The character of the material to be treated, whether ores, concentrates, or tailings, will demand certain modifications of the treatment, without interfering with the principle.

**(a) Percolation of Ores.**—It is advisable to dry-crush the ores; the less dust produced the better for percolation. Screens of thirty meshes to the lineal inch will be found satisfactory in most instances; in some cases a coarser screen may do, or a finer one may be required. It is desirable to crush as coarse as possible without interfering with the percentage of extraction. Stamps are much in use for dry-crushing; mortars with double discharge will give more product and less dust. Rolls are to be preferred, on the ground of their giving a product of greater uniformity. The ore is charged, either directly or through hoppers, into the vats in which the percolation is conducted. These vats, or tanks, may be constructed of wood, brick and cement, concrete, iron, or steel, and vary in size in accordance with local circumstances and requirements. The largest in existence are the circular brick vats at the Langlaagate Estate and Block B Company's works in Johannesburg; these vats have a capacity of about 400 tons, and are 40 ft. in diameter by 10 ft. deep. A size in common use in Johannesburg is 20 ft. in diameter and about 6½ ft. in depth, inside measurement, of which I give Mr. J. S. MacArthur's description: "This vat is made of the best white pine; the staves are 7 ft. 3 in. long, 4 in. wide, and 2½ in. thick, and fitted with a slight taper upwards, so that the diameter at the top is about 4 in. less than it is at the bottom. The bottom is made of the same kind of material, but is at least 3 in. thick. The pieces are fitted together with dowel pins; it is then fitted into a groove cut in the staves about 6 in. above the ground, and the whole vat is bound together by steel hoops with a 6 in. over-lap and at least three rivets. No white lead or packing of any kind should be used in making these vats. If the faces of the wood are true no amount of white lead or other packing can make them truer; if they are not true, neither white lead nor any kind of packing will secure tightness. Besides this the cyanide solution being alkaline would quickly combine with and remove the oil of the white lead, if such were used, and make the vat positively worse than if none had been employed. Circular pieces, cut out of the solid wood and not bent by steam or moisture, are fixed by screws on the bottom of the vat, about 1 in. from the staves, all around the circumference. These circular pieces are about 3 in. thick and 3 in. wide, but the length of each is immaterial, provided always that a complete ring is formed. Wooden slats, about 1 in. thick and 3 in. high, are fixed about 6 in. apart all over the bottom; and an iron pipe, generally 2 in. in diameter, is screwed in from the under side near the center point. The 3 in. space from the bottom to the top of the slats is filled in with round and clean pebbles. Over this surface, formed by slats and pebbles alternately, is stretched



—DETAILS OF THE FALSE BOTTOMS—
—OF THE—
—PERCOLATION VATS—
—SCALE 1/4 INCH = 1 FOOT—

COP. FROM W. R. FELDTMANN'S "NOTES ON GOLD EXTRACTION" ETC.

a canvas cloth to act as a filter, which is fastened by stretching it over a circle piece and ramming the cloth tight by pressing an inch rope into the space between the circle pieces and the staves. The canvas filter is made by shaping and sewing the canvas into a circle piece rather larger than the area of the vat bottom. In practice the canvas filter is often protected by covering it with old sacks or cocoa matting, which serves to protect the filter proper from the wear and tear caused by the friction of the ore or by the cutting of spades." (Special vat and tank constructions will be given aside from this general description in reviewing large and successful plants in different countries.) "The vat thus protected and fitted is charged with ore, and the cyanide solution is run, preferably from the bottom, by a pipe and rises slowly through the crushed ore. It must not be allowed to rush in or rise violently, as by so doing channels will be formed through which the solution will pass without acting on the ore. Such channels are apt to be formed under any circumstances, and should always be guarded against. After the upward percolation, the stopcocks are shut, and opened again after the desired time of contact has passed, so as to allow of a reversal and downward percolation. The cyanide solution now containing gold is carried through the precipitating appliances and from there into the sump, from which it may again be used for percolation. It is not wise to attempt to make the solution very rich in gold, and it is considered better practice to remove the gold frequently, as it is found that a cyanide solution containing gold is not so active as a similar solution without any."

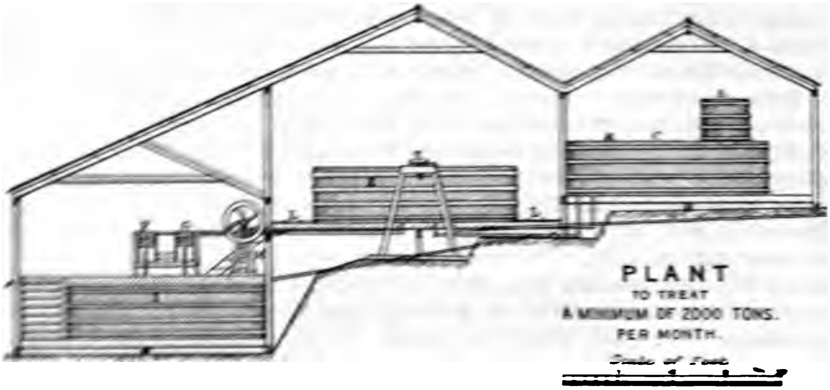
In some instances, however, it has been found of advantage to use gold cyanide solution over again, without first passing the same through the precipitation boxes. (See experiments made by the Robinson Company, in Johannesburg, page 65.) According to the richness of the ore and the fineness of the grinding, percolation may be repeated several times, but after the final percolation with the ordinary cyanide solution, a washing of weak or waste solution should follow, and the whole operation be completed by a water-wash, after which the residues are discharged. The filling and discharging can be done either by hand or by mechanical appliances. The various methods will be described in connection with the process in South Africa.

(b) **Tailings** are treated in substantially the same way as ores, and, the quantities being large and the grade low, the vats of the largest size and the most complete arrangements for saving labor in charging and discharging are necessary for profitable working; generally speaking, the difficulty of discharging the vats is increased by the increase of their diameter and their depth. "Several difficulties arise in the case of tailings, which do not usually present themselves with ores. These difficulties are chemical and mechanical. The chemical difficulties have been described in the chapter on chemistry; no general rules can be applied to them; each case has to be investigated and steps be taken accordingly. The mechanical difficulties arise from the tailings being derived from the operation of wet-crushing. When tailings are charged in a wet state into the percolating vat, they are apt to remain in lumps, from which the water has to be expelled by the cyanide solution before the latter can effectually do its work. It is obvious that where tailings are already saturated with water, the cyanide solution will have a difficulty in penetrating, and this difficulty is increased when the wet tailings

are held together in masses between which the cyanide finds an easier channel for flowing than by soaking through them. This is merely a form of the channels above referred to. Assuming, however, that these channels are not formed, the tailings in a wet state mass or pack together to such an extent as seriously to retard percolation. Another difficulty which arises from the tailings being wet is that in clayey ores the slimy portion of the mass is apt to gather into a layer by itself, which if formed of real clay, not only impedes but absolutely prevents percolation. In order to overcome this difficulty the simple method of drying and mixing should be adopted. The drying is a mere preliminary to the essential of thorough mixing. Particles of clay, which are not kept apart by sand, will agglomerate and form water-proof strata. It is impossible, unless the whole material is perfectly dry, to get the particles of clay separated from each other and allow the sand particles to intervene. Even when this is done, the tendency of the clay particles to agglomerate must be guarded against and prevented. The principal precaution necessary is that the solution, whether applied from top or bottom, should not flow more quickly than the dry tailings can absorb it. In many respects upward percolation has an advantage, but principally because the flow of solution is against gravitation. In downward percolation, where the flow of solution and gravitation act together, the whole material tends to become compressed into a cement, through which the solution penetrates but slowly, preferring to take the easier course down the sides of the vat, and in fact going around rather than through the tailings. Alternate upward and downward percolation may be found useful in some cases."

The percolation vats are charged with tailings to within a few inches from the top, and their surface is leveled. The cyanide solution of, say from 0.2 to 0.8 per cent of strength, is then permitted to penetrate the tailings, till the liquid covers them. The contents of the vat will settle some inches, which shrinkage depends on the depth of the vat and the percentage of moisture in the material. The solution is permitted to remain undisturbed in contact for say twelve hours; after that time it is allowed to drain off. As the liquid is drawn off, it is replaced by fresh solution. This operation is continued for a longer or shorter period in accordance with the value of the tailings (about six to twelve hours in the works of the Robinson Company at Johannesburg). After this time, which is termed the "strong solution leaching," a weaker solution, containing say from 0.2 to 0.4 per cent of cyanide, is turned on, which filters through the ore for about eight to ten hours. This weak solution, when drawn off, is treated separately (see above). At last, water is run on the tailings for replacing the last weak cyanide solution. The volume of solution in constant use and circulation remains the same. The weak cyanide solution is the liquor which has previously passed through the process by which the gold and silver are precipitated and has from the sumps been pumped back into the vat.

The percolation vats, which used to be square, are, in new works, round. The cyanide solution of the described strength has no appreciable deleterious effect either on the wood of the vats or on the iron pipes and iron valves of the pumps. Iron or steel vats may be protected by a coating of coal tar and asphalt, or a solution of asphalt in turpentine, preferably put on hot, if special reasons make such protection desirable. The quantity of cyanide solution used for the treatment of




one ton of tailings amounts generally to half a ton of strong solution and half a ton of weak solution and wash.

When the percolating process is finally completed, the exhausted tailings, or "residues," are discharged in older works by being shoveled out over the side. More modern works have trap-doors at the bottom of their vats for discharge (see diagram, page 59). "Sluicing out" of the residues is being practiced in several localities. The large vats of the Langlaagte Estate Company's works at Johannesburg, which hold 400 tons of tailings each, are discharged by means of running cranes (see diagram, page 57).

It must be always borne in mind that the most complete arrangements for saving labor in charging and discharging large quantities of low-grade material are necessary for profitable working. In order to achieve that end, a plant should offer all such facilities which circumstances permit; it should be so arranged that the tailings will not have to be lifted, but can be dumped into the percolation vats. The size of the vats has been constantly increased. New works, like the Roodeport works at Johannesburg, are supplied with vats 40 ft. in diameter. The table appended illustrates the dimensions of the percolation vats in some of the more important cyanide extraction plants:

*Sizes and Material of Percolation Vats.*

| Name of Works. | Location. | Number of Vats. | Material. | Form. | Dimensions. | Contents in Cubic Feet. | Centents in Tons. | Class of Material Treated. | Time of Operation in Days. |
|---|---|---|---|---|---|---|---|---|---|
| 1. Crown Mines Company | New Zealand | 24 | Wood | Square | 11 ft. by 9 ft. by 3.75 ft. | 371.25 | 7 | Ore | |
| 2. Great Mercur Company | New Zealand | 3 | Wood | Square | 16 ft. by 12.5 ft. by 3.5 ft. | 700 | 20 | Tailings | |
| 3. Tryfluke Company | New Zealand | 4 | Wood | Square | 12 ft. by 16 ft. by 4 ft. | 768 | 20 | Tailings | |
| 4. Waihi Company | New Zealand | 13 | Wood | Round | Diam. 22.5 ft., 4 ft. deep. | 1,589.6 | 30 | Ore | 7 |
| 5. Cripple Creek Gold E. Company | Colorado | 4 | Iron | Round | 20 feet diam. | | | Tailings | |
| 6. Revenue Company | Montana | | Wood | Round | 10 ft. diam., 4.5 ft. deep. | 353.26 | | Tailings | 1 to 1.5 |
| 7. Mitchell Creek G. M. Company | N. South Wales | 6 | Wood | Round | 18 ft. diam., 5 ft. deep | 1,271 | 35 | Tailings | 3 |
| 8. African Gold Recovery Comp'y | Transvaal | | Wood | Square | | | 35 to 50 | Tailings | |
| 9. Crown Reef Company | Transvaal | | Wood | Square | 16 ft. by 18 ft. by 6 ft. | 1,728 | | Tailings | 3 to 3.5 |
| 10. Crown Reef Company | Transvaal | | Brick | Square | 40 ft. by 40 ft. by 10 ft. | 16,000 | | Tailings | |
| 11. Durban-Roodeport Company | Transvaal | | Wood | Round | 40 ft. diam., 7 ft. deep | 8,792 | | Tailings | |
| 12. Nigel Company | Transvaal | | Wood | Round | 16 ft. diam., 5 ft. deep | 1,004 | | Tailings | |
| 13. Nigel Company | Transvaal | | Wood | Square | 16 ft. by 24 ft. by 5 ft. | 1,920 | | Tailings | |
| 14. Langlaagte Estate Company | Transvaal | | Brick | Round | 40 ft. diam., 14 ft. deep. | 17,584 | 400 | Tailings | |
| 15. Robinson Company | Transvaal | 12 | Wood | Round | | 2,000 | 75 | Tailings | |
| 16. Simmer & Jack Company | Transvaal | | Wood | Round | 42 ft. diam., 14 ft. deep. | 23,386 | | Tailings | |

All vats should be some distance above the ground, so that leaking can be easily detected; concrete foundations for the vats are generally adopted. The wooden tank material is an absorbent of both cyanide and gold, particularly when new. It has been found at the Salisbury works, Johannesburg, that pine wood lying thirty-four hours in a 0.3 per cent cyanide solution reduced it to 0.05 per cent, while cement reduced it to 0.24 per cent. Cement tanks have come into use of late, and have proved satisfactory; such tanks and vats may advantageously be built into excavations in solid ground. Many attempts have been made to discharge tailings-pulp direct from the plates, or ore-pulp direct from the mortars into the percolating vats, but their successful treatment by cyanide when so discharged has been prevented by mechanical causes, the reason being that the material packs so densely in the vats that it makes percolation an extremely tedious operation, and in consequence of the presence of slimes the results are unsatisfactory. The advantages of wet-crushing over dry-crushing are, from an economical standpoint, so obvious, however, that experiments will be continued, and ultimately the drawbacks which now adhere to the method may be overcome. Cyanide of potassium solution has been used, in some instances and in an experimental way, in lieu of water in the mortars, when wet-crushing was resorted to, but does not appear to be practiced anywhere at present. An innovation in percolation consists in the circulating system, which will be described in detail in connection with the practice of the cyanide process in South Africa (see page 46).

(c) **Percolation of Concentrates.**—"These are treated similarly to ores, but being generally richer require a greater number of percolations, and thereby a much longer time. In most cases, their quantity is limited, and the size of the percolation vats varies in accordance with the quantity." I have, in most cases, given agitation the preference to percolation for treatment of concentrates, on account of its greater cheapness and rapidity; in Africa most companies prefer the latter method. Percolation of concentrates requires about twenty days, the reason for which will be found partly in the coarser character of the gold, partly in its being in the form of amalgam, and mainly in the difficulty the solution has in penetrating between the faces of the sulphuret crystals. A difficulty sometimes arises in the percolation of concentrates, owing to the crystalline form of iron pyrites and galena. These minerals crystallize in cubes, and when suspended partially or wholly in a fluid tend to range themselves face to face, so that a section of such a mass deposited from a fluid would resemble a brick wall in structure. This difficulty does not arise in the case of sand or minerals which crystallize in other systems. Whenever it occurs, it may be overcome by mixing the cubical sulphurets with coarse sand.

## C. Cyanide and Cyanide Solutions.

The best strength of solutions to use in either percolation or agitation depends entirely upon the nature of the ore, and it is impossible to set any rule. The strength of solutions generally used varies from one eighth to one per cent of cyanide. (In reference to the determination of the correct strength to be used in treating any class of ore, see chapter on laboratory work, page 44.) "For convenience and economy of work,

the solutions are generally divided into three classes: No. 1, No. 2, and No. 3, of which No. 1 is the strongest and No. 3 the weakest. Assuming that the material under treatment does not require a preliminary alkaline wash, or that such treatment has already been completed, it is usual to run on a weaker solution, say No. 2, in the first place, and after its percolation to use No. 1, and then No. 3 in the same manner, finishing with a water-wash, the first portion of which is run into and forms part of the No. 3 solution. These different solutions are kept separate after percolation, and when charged with gold are subjected by themselves to the precipitating process. In some works sumps are used as reservoirs, and the solution is pumped direct from them onto the ore; but space permitting, it is considered better practice to have reservoirs for each solution above the percolation vats, from which the flow can be more easily regulated." For the purpose of bringing weak solutions up to a certain standard, it is advisable to use a very strong solution, of which enough is added to bring the weak solution up to the required strength. This method has to a great extent taken the place of the old method of using solid cyanide to bring weak solutions up to high standards. The strength of the cyanide solutions, which it is of great economical importance to determine, is tested according to Liebig's method by means of a one tenth standard solution of nitrate of silver, which is made by dissolving seventeen grams of pure nitrate of silver in one litre (1,000 cc.) of distilled water. Liebig's method is based on the fact that silver cyanide is soluble in excess of potassium cyanide, with formation of a double cyanide of silver and potassium:

$$\underset{\text{(Potassic cyanide.)}}{KCy} + \underset{\text{(Argentic nitrate.)}}{AgNO_3} = \underset{\text{(Argentic cyanide.)}}{AgCy} + \underset{\text{(Potassic nitrate.)}}{KNO_3}$$

$$\underset{\text{(Argentic cyanide.)}}{AgCy} + \underset{\text{(Potassic cyanide.)}}{KCy} = \underset{\text{(Argentic potassic cyanide.)}}{KAgCy_2}$$

As soon as the whole of the cyanide has been converted into a soluble silver salt, an additional drop of silver nitrate will produce a permanent precipitate of the insoluble simple cyanide of silver:

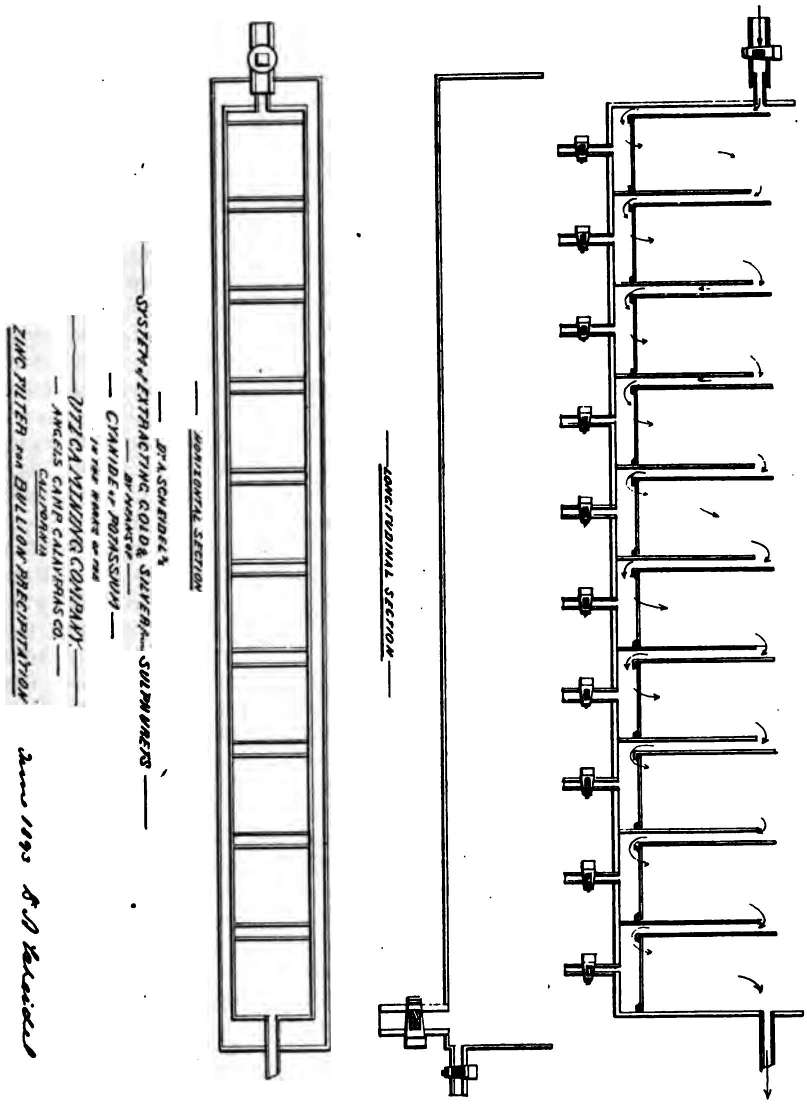
$$\underset{\text{(Argentic-potassic cyanide.)}}{KAgCy_2} + \underset{\text{(Argentic nitrate.)}}{AgNO_3} = \underset{\text{(Potassic nitrate.)}}{KNO_3} + \underset{\text{(Argentic cyanide.)}}{AgCy}$$

A measured portion of the perfectly clear cyanide solution which is to be tested is taken; if necessary some distilled water is added, and the standard silver solution is gradually added from a graduated burette, until a permanent white cloud is formed. As each cubic centimetre of the silver solution is equal to 0.013 grams of potassium cyanide, by multiplying the number of cubic centimetres consumed by 0.013 the amount of cyanide in the solution tested is found in grams, from which the percentage can easily be calculated. A convenient silver solution for the purpose of analyzing cyanide solutions is one of such strength that every cc., added to 10 cc. of the solution which is to be tested, corresponds with 1 per cent pure cyanide of potassium. "The cyanide solutions are apt to form, by continued exposure to the air, carbonate of ammonia; and as this salt interferes very seriously with the determination of the cyanide, it is well to add a few drops of solution of iodide of potassium, which forms a pale-yellow cloud insoluble in ammonia,

on which the finely divided zinc is placed. After passing through the zinc, the solution leaves the chamber at its top; then it descends between the double partition to the space below the perforated bottom of the succeeding chamber, where it undergoes the same treatment, and so forth in all successive ones (see sketches, pp. 31, 33). "This arrangement has been adopted because the gold is precipitated on the zinc in a state of fine division, and would, if deposited on the upper surface, prevent the further flow of the solution; but being deposited on the under surface, the gold precipitate falls off and leaves the passage clear. Each precipitating box may contain ten to twelve double chambers, and no matter how rich the solution is at the inflow, it should not contain more than a few grains per ton at the outflow." In some works the zinc boxes are up to 40 ft. long. There is, however, no advantage in going beyond a limited number of chambers, as precipitation of the metals takes place chiefly in the first few compartments of the box. (See description of Utica plant, p. 89).

I give here Mr. J. S. MacArthur's description of his construction of filter boxes, and the mode of working them which is used by the MacArthur-Forrest patentees in South Africa: "The gold precipitate falls through the gauze 'a' (see cut, p. 31) into a chamber which communicates with an inclosed launder or gutter. From day to day fresh zinc is added, always adding it in the last chamber and bringing the partly consumed zinc up a step, so that the first chamber contains zinc half consumed and rich in gold, while the last chamber contains fresh zinc containing no gold. At intervals of about two weeks, there is a clean-up, and the gold is collected by stirring the zinc so as to cause the gold precipitate to fall off. When this is done, the stopper 'b' is raised and the gold precipitates fall through the opening into the launder 'B.' When this has been completed for each chamber, the launder is discharged through the opening 'C.'" The precipitation boxes are usually made of wood, and although I have been well satisfied with such material (kauri-pine in New Zealand), I substituted it in California by steel, which I found in every respect an excellent material for the purpose. I could not ascertain any increased loss of cyanide by the use of iron as box material; the galvanic action of iron and zinc in contact on the cyanide seems by some writers overrated (see description of Utica plant, p. 89). Of this apparatus, which is based on the same principle, but which in its construction is simpler than the one described, I give the appended diagram. The total length of the apparatus is 9 ft., the size of the chambers is 9 in. by 9 in. by 14 in. deep; the distance of the partitions between each chamber is 1 in. The perforated and movable false bottom of each chamber is of steel, which is an advantage over wire sieve bottoms, which easily become clogged by bullion. The real bottom of each chamber has a faucet of 1 in. diameter, which discharges the liquid and the finely divided bullion into a tank below the apparatus, from where, after settling (under addition of some alum, if saving of time is an object), it is transferred to a vacuum filter, as described further on.

"The zinc used for precipitation purposes should be the best quality found in commerce, and should not contain arsenic or antimony; a small percentage of lead, however, does no harm, but rather tends to promote rapid action by forming a voltaic couple with the zinc" (MacArthur). The metal is preferably used as shavings, or filiform, as these forms




give in practice the most surface for the least weight and do not pass readily through a sieve, whereas the gold, which is precipitated as a fine powder, does. Shavings have the advantage of not forming lumps so easily in the precipitation boxes as filiform zinc; the latter has, however, the greater advantage of being cheaper in its preparation, as no remelting of the commercial zinc is required. It is prepared by cutting sheet zinc into disks, a number of which are placed together and turned on a lathe with an ordinary chisel. The zinc linings of the cyanide packing-cases, for which there is no market, may be turned to account in that way.

In reference to the cost of preparing the zinc, I may quote the Nigel Gold Mining Company in Johannesburg, where one native, working about eight hours a day, can easily keep the works going, with an output of about two thousand ounces of gold monthly; the consumption of zinc is about twenty pounds daily. As a rule, one cubic foot of zinc shavings in the precipitating box is sufficient for the precipitation of the gold from two tons of solution per twenty-four hours, or, roughly speaking, from the same weight of ore (see the zinc for bullion precipitation in Africa on page 53). Zinc in sheets and granulated zinc have been tried for bullion reduction, but with indifferent results, on account of their limited surface. Zinc amalgam and zinc dust have not answered, for mechanical reasons—zinc dust packing too tightly and zinc amalgam not offering sufficient surface in proportion to its weight. The precipitation of the metals in the zinc boxes takes place rapidly; the zinc in the compartment near the influx will be much more quickly charged with bullion than that in those more distant, and the zinc will be consumed in proportion. Zinc on which bullion is already deposited is more active than new zinc; it is therefore advisable to replace the dissolved zinc in the upper by zinc from the lower chambers, and to add the fresh zinc in the last compartment. The generation of hydrogen in the boxes is liable, by polarization, to partly interfere with the bullion precipitation; the zinc in the boxes should be stirred up occasionally to avoid this.

The zinc boxes are cleaned up once or twice a month; for that purpose the inflow of the solution is stopped. The zinc shavings are stirred with a rod, which causes the fine bullion to fall off and to pass through the perforations of the false bottoms, and through the faucets at the real bottoms, into the box below, where it settles readily, on the addition of a little alum. A jet of water will further wash the zinc in the chambers. This method of operating takes only a few minutes, and has been used by me in California. The liquid standing above the settled bullion is returned to the zinc box; the bullion itself, unavoidably mixed with fine zinc, is transferred through a fine sieve onto a vacuum filter. If a final cleaning-up is desired, it will be necessary to dissolve the whole of the zinc, impregnated with bullion, in acid; such necessity will, however, rarely arise. The manipulation itself if required, offers no difficulties. The precipitated bullion is very finely divided, and provision should be made to prevent its flowing away with the liquid out of the precipitation boxes (see page 52).

The process of bullion precipitation by zinc is, generally speaking, a satisfactory one, although not free from objections; all operations with and manipulations of the precipitated bullion require care to avoid loss. The action of the zinc on gold solution is theoretically very simple—a simple substitution of gold by zinc according to the equation:

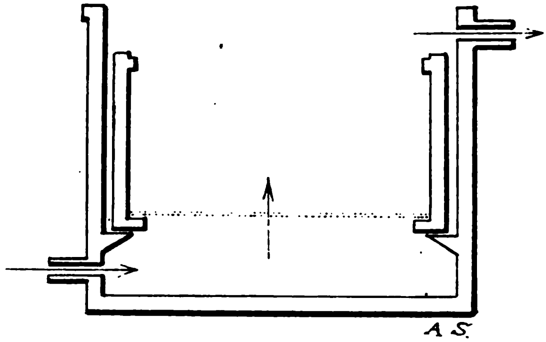
$$\underset{\text{(Auro-potassic cyanide.)}}{2\,KAuCy_2} + \underset{\text{(Zinc.)}}{Zn} = \underset{\text{(Zinc-potassic cyanide.)}}{K_2ZnCy_4} + \underset{\text{(Gold.)}}{2\,Au}$$

One pound of zinc should precipitate about six pounds of gold. The actual consumption is, however, considerably larger, and amounts to from 5 oz. to 1 lb. of zinc per ounce of gold recovered. A constant generation of hydrogen gas in the precipitation boxes proves the effect of the potassic hydroxide on the zinc, and probably a decomposition of cyanide of potassium, going on parallel with the decomposition of the

double gold cyanide. A considerable loss of zinc occurs generally in refining the precipitated bullion, which always contains a high percentage of that metal. The double salt of auro-potassic cyanide appears to be one of the most stable of gold salts; its decomposition by zinc is, however, practically complete; an excess of cyanide of potassium in the solution does not redissolve precipitated gold in the boxes as long as there is zinc present. The cyanide of potassium formed into a double salt with zinc during the gold-reducing process is not available for dissolving gold in new operations. If a surplus of caustic soda has been used for neutralizing acid salts in the ore without following washing, the loss of zinc will naturally be increased. A white precipitate is constantly accompanying the reduction of bullion in the zinc boxes, undoubtedly the result of the action of alkali on the zinc and of the zinc-potassium oxide on the double cyanide of zinc and potassium, which is always present in the solution, forming the insoluble cyanide of zinc; ferrocyanide of zinc is also formed in the boxes. Ferrocyanide of zinc is formed in the percolation vats when the double cyanide of zinc and potassium comes in contact with the iron salts in the ore, and, as it is insoluble, to this cause is due the constant removal of zinc from the solution with the residues (Buckland). The gold precipitate on the zinc is, as a rule, brown to black, with sometimes a metallic luster; it is mostly slimy, and when dry it seldom contains more than 40 or 50 per cent of gold and silver, the remainder being finely divided zinc and its accompanying impurities, such as carbonate of lead. It may also contain copper, if that metal is present in solution. (In the instance of treating concentrates containing carbonate of copper from the Sylvia Mine, New Zealand, the gold solution contained a very appreciable quantity of copper; this complicated matters by causing the copper to precipitate with the gold and cover the zinc, thus forming a galvanoplastic coating, which made it necessary to dissolve the whole of the zinc for the purpose of obtaining the bullion, till I found an addition of cyanide to the solution, before it enters the zinc boxes, useful for the prevention of the deposition of the copper.)

I always found mercury in the zinc bullion in not inconsiderable quantities when concentrates had been treated by agitation; such mercury must have been derived from amalgam, and mercury saved with the pyrites on the concentrators. Gmelin and others describe mercury as absolutely insoluble in cyanide. There is, however, no doubt of the correctness of my observations; the mercury must have been dissolved in the cyanide solution, which entered perfectly clear into the zinc boxes; solubility of gold amalgam in cyanide may offer the explanation. Traces of antimony and arsenic have also been found in the bullion. The precipitation of silver goes on by zinc simultaneously with the gold; it is even more rapid and complete than that of gold (see my table on the process in the Utica Works, page 91.)

Other apparatus than the described boxes have been suggested for bullion precipitation by zinc—for instance, earthenware and porcelain vessels have been recommended. They have the apparent advantage of cleanliness; their construction, however, makes the cleaning-up of the bullion difficult, the connection between the single cells being complicated; they have not become a practical success in works of any extent; they are all based on the principle of the solution penetrating the zinc from below and running off at the top. The precipitates, obtained as




ALMARIN B. PAUL'S PORCELAIN FILTER
FOR THE PRECIPITATION OF
GOLD & SILVER from CYANIDE SOLUTION
BY MEANS of ZINC.
1 INCH = 6 INCH

described from the zinc boxes, are transferred to a sieve, made of No. 1 punched battery screen or a 40-mesh wire screen, through which they are washed onto a filter in connection with a vacuum chamber, where they are liberated from the adhering cyanide solution and reduced from their very voluminous state into a more compact form. This filtration will always be found slow on account of the extremely slimy character of the bullion. For filtering and washing the bullion slimes, filter presses may be suggested. By the screening process the coarser particles of zinc are separated from the bullion, but the bullion still contains a large percentage of very fine zinc, of which it is advisable to remove as much as possible before melting.

*Bullion Refining.*—The means for this purpose are calcination or roasting and acid treatment. I use for roasting (see description of Sylvia plant, p. 79) a muffle furnace, where the slimes are dried, and then calcined for the promotion of the oxidation of the base metals. The calcining process is generally in use in South Africa, and will be described with the cyanide practice in Johannesburg, page 54. I generally prefer sulphuric acid treatment, with following washing and drying of the bullion. The acid treatment is a comparatively simple operation, and does not require, even for large quantities of bullion, any other apparatus than wooden tubs, the increased temperature produced by the reaction of the acid on the zinc making application of artificial heat superfluous. The separation of the acid solution from, and the washing of, the bullion is best done by decantation, and completed on the bullion filter mentioned above. It is advisable to liberate the bullion as much as possible from base metal before melting, which is otherwise connected with loss of gold by evaporation caused by the volatilization of zinc; besides, the zinc fumes are very disagreeable. The presence of oxides of base metals (as obtained by calcination) makes the melting tedious and expensive on account of the detrimental influence of the slag on the melting pots. The presence of a high percentage of base oxides prevents the use of graphite crucibles and compels the use of clay pots. Bullion, when treated by acid as described, does not offer any difficulties in melting, if proceeded with in the following manner:

Melting Room for Cyanide Bullion in Sylvia Company's Works, Tararu, New Zealand.

The bullion, after having been treated with sulphuric acid and washed with water, is dried by suction on the vacuum filter as much as possible, after which it is easily detached from the filter cloth. The mass is then charged into the muffle (see plans of Sylvia cyanide plant, page 77). The heat is kept low to drive off the moisture; it is then increased gradually to dark red; after about one hour's calcination, during which time the oxidation of base metals which escaped removal by acid treatment is going on, the mass presents a gray-brown appearance. Attention has to be paid to the draught to prevent loss of the fine precipitates. Bullion resulting from the treatment of concentrates will invariably give off quicksilver vapors; condensation yielded only a small quantity. The calcination process completed, the roasted bullion is carefully transferred from the muffle, by means of a small shovel, into a wrought-iron box for cooling purposes. It now presents itself in the form of lumps, approaching more or less the spherical form, of the size of peas, largely mixed with dust. When sufficiently cooled, it is charged into a pulverizing cylinder of sheet-iron, 3 ft. long and 2 ft. in diameter, which is revolved by means of a pulley. Large pebbles are charged into the apparatus with the bullion to aid pulverization. Borax, with preference borax glass, and soda ("ammonia-process soda") are added into the barrel, in proportions according to experience, for securing a fusible clear slag of light specific gravity. If the bullion is base on account of a large proportion of zinc oxide, which happens only if acid treatment was not properly conducted, a silicious flux, like sand or glass, has to be added. Acid sulphate of soda and fluorspar have been occasionally found useful as additional fluxes. During pulverization, a thorough mixture of bullion and fluxes will take place. Moisture in the fluxes should be avoided, as it is a certain source of loss in melting, the escaping water carrying fine bullion out of the pot.

Plumbago crucibles are very well adapted for melting bullion, prepared as above described; they stand almost as many operations as with battery gold. In melting, some borax is first put into the crucible; the bullion mixture from the pulverizing and mixing cylinder is not added all at once, but as each portion melts and sinks down, fresh quantities of it are put in. The melting goes on speedily and in the most satisfactory manner. When the whole quantity is finally charged, the temperature is kept very high for some time, to give the small bullion globules a chance to collect. The contents of the pot are poured into a heated mold in the usual manner. No chemical losses will be experienced by this method of bullion melting. Bullion obtained by the described manipulation will be found to be at least 950 fine. The bullion produced by cyanide works generally varies in fineness according to the attention paid to its refining. Gold purchasers buy bullion on assay, and refiners charge higher rates for baser bullion; it is therefore as a rule cheaper in the end to produce clean bullion. Bullion precipitation by means of zinc is not free from objections; its practice is connected with a loss of cyanide and with the introduction into the process of a new compound (the double zinc-potassium cyanide), which is, to say the least, not an advantage; the consumption of precipitating agent (zinc) is far in excess of the amount theoretically required, and no opportunity is offered for a regeneration of the cyanide. The precipitation itself is, however, a very efficient and simple procedure, not requiring either motive power or more than ordinary attention, and the treatment of the

(d) **The Pielsticker Process** applies likewise the electric current as the precipitating agent. A description of this process, illustrated by a diagram, will be found in the patent-specification (see Appendix). No information in reference to practical results could be obtained. The process attained of late a certain notoriety on account of the patent litigation now pending between the owners of the MacArthur-Forrest patent and the Cyanide Gold Recovery Syndicate (Limited) of London, who control the Pielsticker patent.

(e) **The Moldenhauer Process** of bullion precipitation consists in the application of aluminium, or alloys, or amalgam thereof, in the presence of a free alkali. It is claimed that aluminium separates the gold very quickly from the cyanogen solution without entering into combination with the cyanogen, but simply reacting with the caustic alkali which is present at the same time, forming therewith an aluminate. The precipitation of gold by aluminium takes place as follows:

$$6\,AuK\,(CN)_2 + 6\,KHO + 2\,Al + 3\,H_2O = 6\,Au + 6\,KCN + 6\,HCN + 6\,KHO + Al_2O_3$$

and

$$6\,Au + 6\,KCN + 6\,HCN + 6\,KOH + Al_2O_3 = 6\,Au + 12\,KCN + 6\,H_2O + Al_2O_3.$$

The whole of the cyanide of potassium which has been combined with the gold is being regenerated, and the consumption of the cyanide is limited to the loss involved by such secondary reactions as act decomposing. (See "chemistry of the process.") The discoverer of this method of bullion precipitation claims that the quantity of aluminium required for precipitating the same quantity of precious metal, is about four times less than the amount of zinc required to produce the same effect. No results of this process applied on a large scale have as yet been made public.

(f) **The Johnston Process** of abstracting gold and silver from their solutions of alkaline cyanide (United States patent 522,260, see Appendix) consists in the use of pulverized carbon, preferably in the form of charcoal. "The pulverized carbon is placed upon suitable supports so as to form it into filters, through a series of which the cyanide liquid is caused to pass successively, leaving the metal deposited upon the carbon. The gold and silver are then recovered by carefully burning the carbon and smelting the residue with the usual fluxes. By thus employing a series of filters, through which the solution is passed successively, 95 per cent of the precious metal contained in the solution is recovered. When only one filter is employed, only about one fourth of the gold can be extracted."

## V. PERCENTAGE OF EXTRACTION.

The percentage of extraction depends on the character of the ore. As I mentioned before, the process is suitable for many ores which for chemical and mechanical reasons are refractory. The commercial question in the selection of a metallurgical process for treatment of a certain

TABLE SHOWING RE

| No. | Material. | | | |
|---|---|---|---|---|
| | Name of Mine. | Character. | Quantity Treated. | |
| | | | Tons. | |
| 1 | Black Jack, Australia | Concentrates | 5 | P. |
| 2 | Croydon District, Australia | Slimes | 0.5 | P. |
| 3 | Day Dawn, Block & Windham, Australia | Sludge | 280 | P. |
| 4 | Day Dawn, Block & Windham, Australia | Sludge | 224 | P. |
| 5 | Day Dawn, Block & Windham, Australia | Sludge | 30 | P. |
| 6 | Day Dawn, Block & Windham, Australia | Sludge | 145 | P. |
| 7 | Day Dawn, Block & Windham, Australia | Concentrates | 16.9 | P. |
| 8 | General Grant, Australia | Concentrates | 5 | P. |
| 9 | Golden Gate, Australia | Concentrates | 11 | P. |
| 10 | Golden Gate, Australia | Concentrates | 9.3 | P. |
| 11 | Mills United, Australia | Sludge | 90 | P. |
| 12 | Mills United, Australia | Sludge | 154 | P. |
| 13 | Mills United, Australia | Sludge | 206 | P. |
| 14 | Mills United, Australia | Concentrates | 9 | P. |
| 15 | Mills United, Australia | Concentrates | 5 | P. |
| 16 | Mills United, Australia | Sludge | 5 | P. |
| 17 | Mount Morgan, Australia | Ore, ironstone, and kaolin | 1 | P. |
| 18 | Mount Morgan, Australia | Ore, kaolin | 0.3 | P. |
| 19 | Mount Morgan, Australia | Ore, mixed | 0.5 | P. |
| 20 | New Towers Extend., Australia | Ore | 9.5 | P. |
| 21 | New Towers Extend., Australia | Sludge | 10.5 | P. |
| 22 | New Towers Extend., Australia | Concentrates | 21 | P. |
| 23 | New Towers Extend., Australia | Sludge | 37.9 | P. M |
| 24 | New Towers Extend., Australia | Sludge | [illegible] | P. M |
| 25 | Alburnia, New Zealand | Ore | | MacC |
| 26 | Crown, New Zealand | Ore | 203 | MacC |
| 27 | Crown, New Zealand | Ore | [illegible] | MacC |
| 28 | Crown, New Zealand | Ore | [illegible] | MacC |
| 29 | Crown, New Zealand | Ore | [illegible] | MacC |
| 30 | Crown, New Zealand | Ore | 6 | MacC |
| 31 | Imperial, New Zealand | Ore | [illegible] | MacCo |
| 32 | Kenilworth, New Zealand | Ore | | MacCo |
| 33 | Kenilworth, New Zealand | Ore | | MacCo |
| 34 | Kenilworth, New Zealand | Ore | [illegible] | MacCo |
| 35 | Kuaotunu, New Zealand | Ore | 9 | MacC |
| 36 | Red Mercury, New Zealand | Ore | | MacC |
| 37 | Silverton, New Zealand | Ore | [illegible] | MacC |
| 38 | Silverton, New Zealand | Ore | | MacC |

Company's works in New Zealand, for instance, 116 tons of ore, of an assay-value of about $25 per ton, returned after the first month only 75 per cent of the gold, instead of 85 per cent as shown by assay; the returns of the second month yielded 80 per cent, instead of 91 per cent; after the third month the actual results came up to the extraction, as per assay—89 per cent. Similar experiences have been made in the Sylvia Company in New Zealand and elsewhere. It has been recommended to

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## V. PERCENTAGE OF EXTRACTION.

The percentage of extraction depends on the character of the ore. I mentioned before, the process is suitable for many ores which chemical and mechanical reasons are refractory. The commercial qu tion in the selection of a metallurgical process for treatment of a cert

ore has to be considered parallel with the chemical, and that process should be adopted which permits the extraction of the largest percentage of bullion at the lowest cost, and with the least investment of capital. The cyanide process is, for this reason, the best yet discovered for the treatment of the tailings of the South African gold fields, although giving only an average of extraction of about 70 per cent, of which about 60 per cent is recovered (see page 52). No other process gave, at the same expenditure, any results approaching it. The conditions of the Witwatersrand ores are considered particularly favorable for the process, yet the extraction figures are, in most instances, not high. The percentage of extraction in various mills in Johannesburg will be given in the chapter on the process in Africa, page 60. The ores and tailings in New Zealand, where cyanide treatment of dry-crushed ores is carried on extensively, give better results. The Waihi ores, pure quartz, the gold free, but exceedingly fine, the silver in form of sulphides, no sulphurets of base metals, give an extraction of from 85 to 91 per cent of the gold assay-value, the silver returns varying from 43 to 51 per cent. The ore of the Crown mines, which resembles those of Waihi, but containing occasionally telluride of gold, yields on an average 93 per cent of gold and 79 per cent of silver. Concentrates, if satisfactory at all in cyanide treatment, give as a rule very high figures. A considerable quantity of concentrates from the Sylvia Mine in New Zealand, of a very complex character, being composed chiefly of zinc-blende and copper pyrites, with a large percentage of galena and iron pyrites, were treated by me by cyanide, and gave very satisfactory results under conditions where no other means of treatment were at disposal. The said concentrates are classified by the dressing plant; the fine slimes rich in bullion and galena gave as high an extraction as 95.43 per cent of the gold and 86.69 per cent of the silver; coarse concentrates gave an average of 80.32 per cent of the gold and 50 per cent of the silver. A large parcel of very fine sulphurets (from the canvas plant) from the Utica Mine, California, consisting of pure iron pyrites in finest division, mixed with more or less fine sand and carbonate of lime, proved an excellent material for cyanide treatment; the extraction averaged 93.18 per cent of the gold value, rising in some instances as high as 96.57 per cent. The coarse concentrates from the Frue vanners did not give such good results, if treated direct; their reduction to greater fineness, however, improved results. An appended table shows the results of successful treatment of parcels of ores from various sources. It is to be regretted that no corresponding table, giving a like description of ores treated with unsatisfactory results, can be produced for comparison, which would be useful and instructive.

The recovery of the bullion should correspond with the extraction shown by assays; in practice, however, there is often a discrepancy, which may be explained by various causes; new vats, particularly those of wood, absorb both gold and cyanide, and considerable differences in the returns will be felt during the first weeks of their use. In the Waihi Company's works in New Zealand, for instance, 116 tons of ore, of an assay-value of about $25 per ton, returned after the first month only 75 per cent of the gold, instead of 85 per cent as shown by assay; the returns of the second month yielded 80 per cent, instead of 91 per cent; after the third month the actual results came up to the extraction, as per assay—89 per cent. Similar experiences have been made in the Sylvia Company in New Zealand and elsewhere. It has been recommended to

soak the wooden parts of a new cyanide plant with paraffine to prevent absorption; a coat of asphalt dissolved in bi-sulphide of carbon will be found a good preventive for the absorption by wood. The chief sources of chronic losses are to be found in the imperfect separation of the gold solution from the exhausted ore residues, and in the faulty methods of dealing with the bullion after its precipitation by zinc. There is no reason why the actual returns should differ from the returns as established by assay, provided all mechanical losses are prevented. In reference to the losses in the Johannesburg mills, see chapter on the process in Africa, page 52.

## VI. WORKING COSTS OF THE PROCESS.

As may be deduced from the whole tenor of this paper, the working costs of the cyanide process vary within wide limits and depend on many circumstances. Locality is a prime factor in the costs of working any process, and expenses must be high where operations have to be carried on in an inaccessible situation, or where there is dearth of fuel, water, building material, etc. Apart from the question of locality, the cost depends principally upon three factors:

The nature of the ore.

The price of labor.

The price of cyanide.

When an ore contains acid salts and demands an alkali treatment, the price of the alkali must necessarily be added to other costs; and where the ores are slimy, recourse must be had to drying and mixing appliances, which also increase the cost to an extent depending on local circumstances. The principal labor involved in the process is the charging and discharging of the vats, or, if agitation is used, the charging of the agitator and the removing of the exhausted material from the filter appliances. The charging and discharging of the percolation vats may, under ordinary circumstances, be contracted for at a rate of about 25 cents per ton; the extent of the operations is naturally of great influence in regulating the cost of handling. Very large works apply mechanical means for discharging the vats, such as dredges and movable cranes, which reduce the expense of labor per ton of ore to a minimum. To give an instance of the labor employed in working the agitation process I mention the Utica works in California, where the handling of the ore and all the labor connected with the treatment amounts to $1 per ton; this applies to concentrates varying from $50 to $95 in value per ton. The cost of the cyanide is one of the principal charges in the process, and the cost of treatment depends to a great extent upon the price of cyanide and on the amount consumed per ton of ore. The price of cyanide of from 95 to 98 per cent strength now averages about 50 cents per pound, delivered at seaports, and for lower strength the rates are somewhat better.

The amount of cyanide consumed per ton of ore is between 1 lb. and 3 lbs.; the character of the ore has, however, the greatest influence on the consumption, and in many cases the cyanide process will be found the best, cheapest, and quickest method, even if a considerably larger amount of cyanide is consumed per ton. Naturally, as the quantity treated is greater the cost becomes proportionally less. In some mines, as at the Primrose

Company, Johannesburg, tailings are treated for about $1 per ton, and it is very seldom that the cost in the Transvaal exceeds $2 50. In the Crown Reef works the cost of treatment ranges from $1 to $1 37 per ton; this includes the royalty, which for the use of the MacArthur-Forrest patents amounts in South Africa to $1 25 per standard ounce of gold. In Revenue, Montana, the cost of treatment per ton of ore, including crushing, amalgamating, cyanide treatment, and royalty of $1 per ton, amounts to from $4 to $5 per ton. The cost of treatment of ore at the Mercur Mine, Utah, amounts to $2 40 per ton, not including royalty. The cost of ore treatment in the Crown mines, New Zealand, is from $3 37 to $3 50; in the Waihi Company's works, New Zealand, the cost amounted to $2 25 per ton of ore; the expenses are now reduced to $1 25, cost of crushing and patent-royalty of 7½ per cent on the bullion value not included. The treatment of concentrates is, as a rule, more expensive than that of ore and tailings; their value is, however, in most cases, considerably higher than that of those materials, so that the cost per ounce of gold extracted is, with concentrates, generally much lower than with ore and tailings. The agitation treatment of concentrates (sulphurets) costs in the Utica works, California, from $3 25 to $3 50 per ton, labor included. "Ores yielding upwards of 90 per cent of their gold assay-value have been treated at $1 25 per ton, and tailings containing less than $3 have been worked profitably. It is therefore safe to assume that under favorable circumstances, and apart from all costs of mining and crushing, the cyanide process is capable of application at a low figure."

## VII. COST OF CYANIDE PLANTS.

The cost of cyanide plants varies naturally with the system applied and the extent of the works. A well-equipped plant with a capacity of 50 tons per day will cost about $25,000, a 100-ton plant about $40,000. I shall have occasion to give details on the cost of plants when describing prominent and successful plants in different parts of the world, and I refer particularly to the corresponding chapter of the process in Africa, page 55.

## VIII. MACHINERY AND APPLIANCES.

In discussing the various methods of applying cyanide, the machinery for each purpose has been described (see Chapter IV). Generally speaking, all plants have the same main features; modifications, however, will be suggested by special conditions of locality and character of ore. Economy in handling the ore is of the greatest importance, and should be made of first consideration in selecting the site for the plant and in its arrangement. The crushing machinery, if ores are to be treated without previous amalgamation, should be selected in accordance with the character of the material. The proper preparation of the ore is a very important item, and the crushing machinery should be selected so as to produce the smallest amount of dust if dry-crushing and of slimes if wet-crushing of the ore is practiced. For dry-crushing, rolls should be preferred, on account of their giving a product of greater uniformity than stamps, which are now used to a great extent. It is with the cyanide process as with other leaching processes, the more

equal in size the particles are, the better. Wet-crushing, with the improvements necessary for mastering the slime difficulty, may ultimately win. Pumps of all constructions may be used for the transportation of the solutions, provided their material is not attacked by the alkaline cyanide solution. I shall refer to the general arrangement of plants in the corresponding chapter of the process in Africa, page 55.

## IX. LABORATORY WORK.

Exact laboratory experiments must precede all cyanide mill operations; the required fineness of the ore, the strength of the cyanide solution, and the length of time for its action on the ore, have to be established by experiment. The correct strength necessary for treating any class of ore may be readily determined in the laboratory by treating a weighed quantity of the ore with cyanide solutions of different strength and for various periods of time. After treatment, the amount of gold extracted and the quantity of cyanide consumed should be determined. These results are then compared with the original contents of gold and silver in the ores and the original strength of cyanogen in the solution used. A method of rapidly determining the gold in the cyanide liquors consists in evaporating a known quantity to dryness on lead foil (free of silver), and cupelling the lead in the usual manner. In the presence of base metals, the liquor should be evaporated with the addition of litharge, and the residue assayed for gold and silver in the usual manner. The point to be aimed at is to consume as little as possible of cyanide and to extract at the same time as high a percentage of gold and silver as possible. The water used for making up the solutions should be examined for carbonic acid, free sulphuric acid, and sulphates. A chief point to be investigated in the laboratory is the "acidity" of the ore, by which term is understood the presence of products arising from the decomposition of sulphurets. These chiefly consist of free sulphuric acid and of products derived from a more or less advanced oxidation of pyritic matter, such as proto- and per-sulphates and basic iron salts. The exact amount of free acid contained in an ore sample can be readily determined by shaking a certain weight with water, and adding standard normal or one-tenth normal caustic soda solution, till the point of alkalinity is reached, as indicated by litmus or some other indicator. The means to prevent the ill effects of acidity have been discussed in the chapter on chemistry. The amount of soda or lime required for the ore is easily calculated from the consumption of normal soda solution as shown by the above experiment. *The cyanide solution will in many cases contain after the treatment of ore the evidence of secondary reactions, and a complete chemical analysis of the solution, before it comes into contact with the zinc, should in all cases be made. The explanation of unsatisfactory results with cyanide treatment will be found in many instances by means of such an examination.*

The reasons why one ore yields its gold readily to cyanide treatment, and others of a similar chemical composition do not, are not always apparent; chemical analysis and microscopical examination should be used along with practical tests. As a programme for the examination of an ore in reference to its fitness for cyanide treatment, the following may be suggested: (1) The ore shall be crushed and passed through a 30-mesh

sieve, and part of it assayed. (2) The "acidity," if any, of the ore, shall be determined, as described before, in say 100 grams; if necessary, water and alkali washes will then be applied before the ore is submitted to cyanide treatment. (3) Shaking tests in glass bottles with solutions of various strength during various periods of time will determine the amount of cyanide consumed by the ore, the strength of solution required, and the time necessary for the reaction. Cyanide determination in the solution and assay of the ore, after treatment, provide the required data. Agitation tests will decide if an ore is suited for cyanide or not. In cases of successful cyanide treatment by shaking tests, percolation tests of samples should then be made. The simplest apparatus for the purpose is a glass funnel of large size, the neck of which is closed by an india-rubber pipe and clip; a lamp chimney closed at one end by a stopper of india-rubber, which carries a glass tube, may be used for the same purpose. The ore is placed on a filter-bed of pebbles, which is covered by a piece of flannel or filter-paper. Experiments with equal weights of ore, but with solutions of various strength, and exposed during various periods of time, should be made simultaneously; analyses of the percolated solutions and assays of the well-washed residues will then show the best conditions of treatment. Small tests, if properly conducted, are excellent guides as to the treatment of ore on a large scale. In a cyanide plant there is ample employment for the chemist in charge; all operations should be controlled by him by assays and analyses; improvements on all parts of the process may be effected by careful observation and continued investigation. The non-success of the cyanide process in various mining camps may be attributed to the incompetency of the men to whose care the works were intrusted. Mine owners will find it profitable to employ, at least during the first time of their using cyanide, a competent man, instead of "finding out" for themselves at the expense of much time and money. When the process is once successfully and firmly established, an intelligent foreman will easily acquire the necessary knowledge to test and make up the solutions.

## X. DANGER IN WORKING THE PROCESS.

The deadly poisonous character of the reagent was once considered to be a great obstacle in the way of its successful introduction. However, in the first place, the solutions used are so diluted that the hydrocyanic acid evolved from them is of no consequence if the works are properly ventilated. For safety, as much as for practical reasons, all materials should be tested for acid before treatment, and, if necessary, neutralized. In the second place, there is no necessity for those working the process to come into contact with cyanide, solid or in solution; even properly conducted cleaning up does not require contact. Some, but very few, have a susceptibility for cyanide, and with them the most diluted solutions, if brought into contact with the skin, produce eruptions, which, although not dangerous, are itching and annoying; such men should not be employed in cyanide works. In case of bad ventilation, complaints of headache, faintness, and dizziness may be heard. In instances where circumstances require that hands and arms should be brought in contact with the solution, I found a coating of oil or coal oil (kerosene)

an effective protection for the skin. The process is now used so generally, and on such a gigantic scale, that many hundred men are constantly employed in the works; considering the dangerous character of the cyanide, the number of accidents is remarkably small, and it may justly be said that no more danger is incurred in the working of the cyanide process, under ordinary precautions, than there is in working in establishments of chemical and metallurgical industries, where corrosive liquids and acids are constantly in use. Extraction of gold and silver by cyanide will compare, as to fatal accidents, very favorably with most chemical and metallurgical industries. It is, however, always well to instruct the men how to act in cases of emergency: Put the patient into a hot bath and apply cold water to his head and back. In cases of internal poisoning, vomiting by physical means, or by emetics, is advised. Freshly precipitated carbonate of iron, obtained by mixing equal quantities of sodium carbonate and ferrous sulphate, is recommended for internal use; the two chemicals should be kept on hand ready for compounding this antidote. If the poisoning is the consequence of inhalation of hydrocyanic acid, it is advisable to make the patient inhale a small quantity of chlorine gas, ammonia, or ether; rubbing with camphor-alcohol is recommended. It has been reported of late that Dr. Johann Antal, a Hungarian toxicologist, has notified the Hungarian Society of Medicine that he had found a perfect antidote for prussic acid in nitrate of cobalt; he quoted forty cases of its use with perfect success.

## XI. EXEMPLIFICATION OF THE PROCESS—THE PROCESS IN VARIOUS COUNTRIES.

After having given the outlines of the process, and the various methods of its use, I now propose to describe its practical application in different parts of the world.

### A. The Process in Africa.

The cyanide process has found its earliest application on a large scale in the Witwatersrand gold fields of Johannesburg, South African Republic. The use of the process has since been extended to the different gold fields of the republic, the output of which has gone on steadily increasing, rather by application of improved machinery and the MacArthur-Forrest recovery process, than by the opening up of new mines. In the following notes, describing the methods of cyanide treatment in South Africa, I give chiefly the general information obtained from Mr. J. M. Buckland, the General Manager of the African Gold Recovery Company of Johannesburg, and embody in them, at the same time, the special information obtained from other gentlemen in charge of prominent companies and works.

*The Ores.*—"At the Witwatersrand, from which at present some forty-five thousand ounces, or about 90 per cent, of the gold obtained in South Africa by the cyanide process is obtained, the ore may be classed generally as conglomerate, locally called 'banket' (almond-rock). This consists practically of quartz pebbles embedded in a quartzose matrix, which last carries the gold and a varying proportion of iron pyrites. There is also a small quantity of alumina in most 'banket,' existing in diverse forms of

corundum and (combined with silica) of clay. The latter is the cause of much of the slimes formed during the crushing process, the cheap and profitable treatment of which by cyanide still remains a difficult problem. The iron pyrites almost invariably contain traces of copper and nickel as sulphides. Within 50 to 100 ft. of the surface, the 'banket' is more or less weathered, and the iron exists in an oxidized form; but below this depth the ore becomes paler in color, less friable, and consequently more difficult to crush, and iron pyrites takes the place of the iron oxide. The gold also is more difficult of extraction by amalgamation or by cyanide; but whether in this latter case the cause is chemical or mechanical is not yet determined. On the latter hypothesis, the difficulty would consist in the fact that the particles of gold are so imbedded between the cleavage planes of the pyrites as to prevent to a great extent contact with cyanide solution; and, on the former, that the gold exists chemically combined with certain constituents of the ore, such as sulphur and arsenic—the cyanide having presumably not sufficient affinity to act as a solvent by breaking up the bond of union.

"At the Sheba Mine, De Kaap District, the ore is a quartzite, containing a small amount of iron pyrites and talc. When crushed, there are formed, owing to weathering, sulphates of iron and free sulphuric acid, which react on the talc (silicate of magnesia), forming sulphate of magnesia, and this salt may be observed as an efflorescence on many of the tailings-dumps. This compound, like all others in which the 'base' is weaker than potash and the 'acid' stronger than hydrocyanic acid, causes decomposition and loss of cyanide (see chapter on chemistry). At the Barrett Company (Kaapsche Hoop), the ore is a very soft, decomposed talcose slate, containing a large proportion of hydrated iron oxide. Owing to the very fine nature of the gold the solution of it is almost complete, but sufficient of the very finely divided portion of the ore passes through the filter cloth, coats the zinc, and prevents thereby the gold solution from coming into contact with it; precipitation is consequently seriously interfered with. The difficulty in such cases is met to a certain extent by the use of an extra vat, placed between the leaching vats and the precipitating boxes, in which the greater part of the suspended matter is allowed to settle, and tolerably clear liquid is filtered off. This ore is of so soft a nature as not to require any explosives in mining; it does not even require preliminary crushing for the purpose of preparing it for treatment, but is simply sifted through drums, and the portion which passes through the sieve, when mixed with coarse tailings to assist filtration, is ready for cyanide treatment.

*Ore Reduction.*—"At present, with the above-mentioned exception, all ores in the Transvaal gold fields, which are treated by cyanide, have previously undergone the ordinary crushing and amalgamation process in the battery. The wet-crushing stamp-mill is the only machine employed for ore reduction, apart from various grinding machines, such as the Wheeler and Berdan pans, which are rarely used for the finer reduction of tailings and concentrates. At the Rand (Witwatersrand), the average mesh employed for battery screens is thirty holes to the linear inch (No. 30 screen), or 900 to the square inch. In some cases, however, No. 24 screens are used. At mines, such as the Sheba, where the gold is very fine, No. 40 screens are employed, but the tonnage per stamp per twenty-four hours is in such cases not much more than two

tons, as compared with the three and two thirds tons usual for mills crushing Rand 'banket.'

"In an experimental way, a solution of cyanide has been used instead of battery water; at present, however, water is invariably used in the mortar boxes, sufficient success not having attended the other method. The use of cyanide solution in the mortars would be of advantage only when the pulp is directly delivered into the percolation vats; the formation of slimes is fatal to this method. There is now no other process than the cyanide process used in treatment of tailings, and no mill of which the tailings are of sufficient value is considered complete without a cyanide plant. On the Witwatersrand alone there are upwards of forty cyanide plants in operation, and ten in process of construction; the present quantity of tailings treated monthly is about 250,000 tons. On the other gold fields—DeKaap, Lydenburg, and Klerksdorp—there are in all ten cyanide plants in operation.

*Method of Applying Cyanide.*—"The method of applying cyanide to gold extraction is that specified in the MacArthur-Forrest patents, of which the African Gold Recovery Company holds the rights in Africa, and lets out the right of using the process to gold-mining companies on royalty, usually 10 per cent on the value of the gold produced. Nearly all companies using the process erect and work their own plants, paying royalty as above." Some custom-works will be mentioned later on.

The mode of procedure in applying the cyanide process in Africa is generally that which has been outlined in the earlier part of this paper. Some further details, however, will here be given and may prove of interest. The general system of treatment is that of percolation in tanks.

*The Vats.*—"The percolation tanks, or leaching vats, vary greatly in size and shape. Those first constructed were from 15 ft. to 20 ft. square, and from 4 ft. to 5 ft. deep, the material used being boards 9 in. wide by 3 in. thick and as long as the side of the vat. Owing chiefly to the difficulty of making these water-tight, oval and finally round vats, composed of staves 2½ in. thick, 6 in. wide, and varying in length from 5 ft. 6 in. to 11 ft., were used. Vats now vary in diameter from 15 ft. to 40 ft., and in capacity from 30 to 600 tons. The employment of bottom discharge, by which the exhausted tailings, or 'residues,' are shoveled through a hole in the bottom directly into a truck below, has rendered the great increase in depth possible. 'Side discharge' through doors in the side of the vat is also employed to some extent. Discharge over the side into trucks on tram-lines is now used only in cases when first cost, or 'want of fall,' in the dumping-ground is a serious consideration. The loading into trucks does not cost more than 4 cents per ton, even under unfavorable conditions, and is only 2 cents and even less per ton in some cases. At the Barrett Company, the residues are shoveled through an opening in the bottom of the vat into a launder, where a stream of water carries them away." Some companies, like Le Champ d'Or French Gold Mining Company, let the filling and emptying of their cyanide tanks by contract.

"In some of the largest works cement vats are used, particularly for 'sumps,' or tanks, where the solution is stored after passing through the precipitation boxes. Such vats are really excavations lined with bricks, laid in hydraulic mortar, and plastered inside with cement; these attain a capacity of 600 tons, being 50 ft. in diameter and 9 to 11 ft. deep. For discharging them, when they are used as percolation

vats, either tram-lines are laid down along the bottom, passing out through doors, which are bolted and made water-tight when the vat is in use, or else trucks are lowered by a steam crane into the vat, and filled by natives, and again hoisted" (see diagrams, page 57). "At the Salisbury works a tailings wheel lifts the pulp, after passing over the amalgamating plates, to a flume, which carries it to a hydraulic separator, which separates the slimes from the coarse tailings. This is done to render the subsequent treatment of the tailings more economical, as easy filtration is achieved when tailings free from slimes are treated. The slimes are treated by themselves and filtered in filter-presses. Many companies now run the material from the mill into intermediate settling-vats, provided with bottom or side discharge, for convenience of loading the trucks which transport it to the cyanide works. The difficulty of insuring an equal mixture of fine and coarse tailings is met by means of a rotary distributor, pivoted above the center of the vat and discharging into it. This distributor consists of pipes of different lengths diverging from a central basin into which the pulp is delivered. The openings at the end of the pipes are so arranged that the stream of pulp issuing therefrom causes the distributor to rotate." In the Nigel Company's works the slimes are separated from the sand by means of tailings-pits, with overflow into pits where the slimes are collected; these are dried, broken up, and delivered, mixed with the clean sand, into the vats in the works.

*Treatment of Concentrates.*—"Concentrates are not now treated by agitation in the Witwatersrand gold fields; percolation has been substituted for it. A period of contact and percolation, extending from two to four weeks, is now usually employed. Agitation has been abandoned on account of its cost, consequent upon the power and constant attention required, and the necessarily small amount treated at one time. In addition, it was found that the cyanide consumption was usually increased by the solution becoming heated, owing to the friction of the solid particles during an agitation of several hours. In treating by ordinary percolation, the concentrates are usually mixed with a sufficient amount of coarse tailings to insure filtration. Transferring the material from one vat to another at intervals of a few days is sometimes considered beneficial, for the purpose of obtaining a supply of oxygen as required by Elsner's equation." (See chapter on chemistry, p. 16.)

"Although pyrites themselves consume practically no cyanide, the great difficulty incurred in the treatment of concentrates generally, and of some tailings, is due to the fact of their having been partially oxidized by exposure to the air." The reactions thereby taking place have already been mentioned as detrimental in the chapter on chemistry, and the remedies enumerated (see p. 18). "The water-washings employed to remove the 'acidity,' as it is termed, take place in a special vat, as the traces of the cyanide retained by the filter-cloth, etc., of the regular leaching vat are liable to dissolve gold, and thus cause loss. Experiments have shown that a water-wash pure and simple will dissolve out of most tailings a minute quantity of gold, but this is so small an amount that it may be neglected. If the ore is not very acid a solution of caustic soda is run on after the last water-wash, and the air contained in the solution will serve to convert the ferrous hydrate formed, which would otherwise subsequently form potassic ferrocyanide

with the potassic cyanide, into innocuous ferric hydrate. If very acid, however, aëration, by changing to another vat, will be necessary.

"Lime sprinkled on the surface of the charge of ore, or mixed with it, is often preferred to caustic soda, and has the advantage of clarifying solutions from organic compounds, which, if present, cause 'frothing' in the zinc boxes. Lime does not form the yellowish-white precipitate in the zinc boxes, which is mainly ferrocyanide of zinc, and liable to occur when caustic soda is used, and which, by coating the zinc, interferes with the proper precipitation of the gold."

*The Advisability of Ore Concentration.*—"The question as to whether it is advisable to concentrate, or to allow the pyrites to remain with the tailings for subsequent cyanide treatment, is at present under discussion, and, like all other matters in gold extraction, resolves itself into a question of cost. The general rule is, that with high-grade pyrites concentration does pay, but with low-grade not." The Nigel Company has abolished the use of concentrating machinery. They found that the extraction of gold by cyanide treatment is equally as good from the ore from which the concentrates have not been taken, as it was when using Frue vanner concentrators. Their tailings are consequently of a high grade, containing pyritic matter, and solutions of greater strength are used than is the practice with other works on the Rand. (W. A. Radoe.)

*The Cyanide Solutions.*—"The strength of the solution before treatment was some four years ago ½ and 1 per cent, but now 0.25 and 0.3 per cent may be taken as the usual amount of pure cyanide of potassium contained in what are usually called strong solutions in South Africa. In most works a constant quantity of this strong solution is run on each charge of ore, having been made up from the 'weak' or dilute solution in stock by addition of a sufficient quantity of the solid salt, or a concentrated solution prepared from it. Should the cyanide consumption of the ore increase, the strength of the dilute solution, or that which has been already used, decreases, and more solid cyanide is required, and *vice versa*. Another method consists in always adding the same amount of solid cyanide to the same amount of weak solution, and in case the latter is below a certain point (say 0.1 per cent) to continually use a larger quantity of strong solution for running on the ore until the 'weak' rises to 'normal' once more. If the strength of the stock solution falls too low, the precipitation of gold is imperfect, probably because the cyanide of zinc formed in the precipitation boxes is not dissolved and coats the zinc. If, on the other hand, it is too high, the consumption of cyanide and zinc, by dissolution of the latter in the former, is unnecessarily great. In this case, too, the loss of cyanide by atmospheric decomposition is increased, and, while the same absolute amount of solution is lost by leakage and in the form of moisture adhering to portions of the residues, yet, the solution being stronger, more potassic cyanide is lost. The quantity of strong solution employed per charge of ore varies according to whether a preliminary washing with a dilute solution has been employed or not. In the former case, it is about 25 per cent of the weight of the ore, and in the latter case about 40 per cent, which last quantity is usually sufficient to just cover the charge. The amount, however, varies in different works, and, within reasonable limits, it is not a matter of great importance, provided sufficient solid cyanide is added daily to keep the stock of weak solution at the right strength. It is desirable that the strong solution be of uniform strength throughout the

whole charge of ore, and this object is attained, in great measure, by using a preliminary washing with a dilute solution; the cyanide of the latter satisfies most of the components of the ore, which consume cyanogen, so that the strong solution, which follows, is free to act on the gold alone. This preliminary wash has also the advantage of saturating lumps of slimes which may be in the tailings and would absorb the strong solution, which would be lost when the residues are discharged. Generally speaking, a larger quantity of the weaker solution is preferable to a smaller quantity of strong solution, but exigencies of time, capacity of the plant, filtering properties of the material, etc., cause modifications of this rule. When a charge has had more than its own weight of washings passed through it, it becomes a question whether there is sufficient increase in yield by continuation of the process to cover the cost of pumping, apart from the fact that if the solution be run too fast through the zinc boxes not only is the gold it contains imperfectly precipitated, but that already deposited is liable to be mechanically carried into the sump by the force of the current."

The strength of cyanide solutions used in the Crown Reef Mine works varies from 0.05 per cent to 0.35 per cent; they range after treatment from nothing to 0.33 per cent. The total quantity of solution used, inclusive of water-washes, is about 80 per cent of weight of charge; extraction takes from forty to fifty hours. (G. E. Webber, Jr.) In the Nigel Company, where, as already stated, concentration has been abandoned, the conditions are in consequence somewhat different from those usually prevailing in cyanide works; for tailings over 24 dwts. (about $18) in value a solution of 0.6 per cent is used, preceded by a weak wash of solution of 0.15 per cent and followed by two weak washes; the liquors drain off at a strength of from 0.4 to 0.25 per cent of cyanide, the first solutions draining off at a lower percentage than the last. The amount of cyanide used per ton is about 3.8 or 3.5 lbs. per oz. of gold recovered (3.5 lbs. of 76 per cent cyanide). (W. A. Radoe.)

*Precipitation by Zinc.*—The precipitation process going on in the zinc boxes has been fully discussed in a former part of this paper.

"After passing through the zinc box the solution should not contain more than 50 cents of gold per ton, and in the majority of cases there will be only a trace of gold present. If appreciable quantities of gold remain unprecipitated, a certain amount is daily lost in the dilute cyanide solution contained in the residues (see above). The latter should be periodically tested for gold soluble in water, and gold soluble in cyanide solution; imperfect precipitation will be discovered by the first, and too short time of treatment by the second test."

*Time of Treatment.*—"The total time employed in the treatment of a charge of tailings varies from three days to a week, and is dependent, from a chemical point of view, upon the greater or less fineness of the gold; the general rule is that the longer the time the better until the increased cost of treatment more than counterbalances the improved percentage of extraction."

*Cyanide.*—The cyanide usually employed contains from 70 to 80 per cent of pure potassic cyanide, but another quality, containing upward of 95 per cent, imported from Germany, is also used, and is preferable for the reasons explained in the chapter on cyanide (p. 30). The consumption of cyanide is about 150 tons per month in the Witwatersrand mines. Germany has sent out nearly 1,000 tons to the Transvaal this year.

*Value of Rand Tailings and Percentage of Extraction.*—"The average value of Rand tailings per ton, before treatment, is $5; of this $3, or 60 per cent, is actually obtained by the cyanide process, $1 50, or 30 per cent, is left in the residue, and 50 cents, or 10 per cent, is unaccounted for. Ores from DeKaap, containing a large amount of mispickel, gave only an extraction of 9 per cent, but on roasting the extraction rose to 83 per cent, the arsenic being presumably driven off. Owing, however, to the cost of fuel, and the high cyanide consumption resulting from sulphates formed by partial oxidation, roasting is never employed as preliminary to the cyanide process. The percentage of gold extracted varies in different localities, but is usually between 70 and 80 per cent. It depends chiefly upon the degree of fineness of the ore, and the degree to which the gold it contains has been liberated from the matrix and exposed to the action of the solution. (In comparing an ounce of like particles of $\frac{1}{30}$ in. diameter with an ounce of particles of similar shape but $\frac{1}{40}$ in. diameter, the surface exposed by the first lot is three fourths of the surface exposed by the second lot—this being a particular instance of the general law, that for equal weights of similar particles the surface exposed varies inversely as the diameter.) There is little doubt that the remaining 20 to 30 per cent in the residues consist of particles of gold still incased in the matrix, and this is proved by the fact that finer grinding renders almost complete extraction possible. The limits of fine grinding on a working scale are fixed by the increased difficulty of filtration. Even when only a No. 30 screen is used in a wet-crushing stamp-mill, it is not possible to filter the pulp in its entirety, on account of the slimes; the consequence of this is that with moderately fine crushing only the coarser portion (possibly 80 per cent) of the tailings is at present treated by cyanide. Slimes of sufficient value are in some cases treated by drying, crushing, and mixing with a sufficient amount of coarse tailings to allow filtration. A mixture of equal parts of each take at least a week for treatment. The drying is performed either by exposure to the sun, or, especially in the case where much organic matter is present, by slightly calcining in a reverberatory furnace, or in form of bricks in a kiln. In the last two cases, however, the cyanide consumption is increased by the oxy-salts of iron formed, and although usually remarkably good results may be reckoned upon, yet the cost involved in so much handling is so high as to be prohibitive for low-grade slimes." The treatment of slimes or of tailings-pulp containing a high percentage of slimes, is still one of the unsolved problems of the cyanide process; at present most of the slimes are washed away, and with them a large amount of gold is lost in Johannesburg.

It has been suggested to solve the difficulty of slime treatment by mixing the slimes with 50 per cent of their weight of a solution containing cyanide of potassium and the double cyanide of manganese and potassium. This mixture is pumped into a filter press under high pressure; after filling the press, water is forced through, washing out the gold solution. An extraction of 97.6 to 98.2 per cent is claimed. (W. Bettel's process. E. & M. J.)

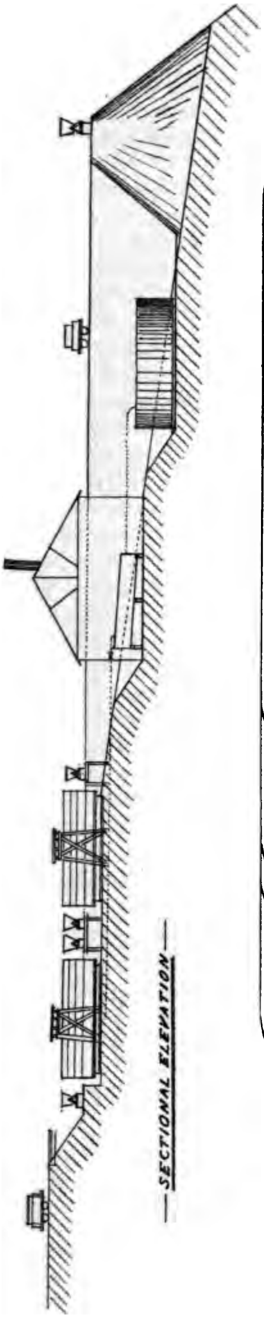
*Loss of Gold, and its Causes.*—"From leakage, and the loss consequent upon the handling of gold slimes in the various stages of conversion into bars of bullion, there is a certain amount of loss; but this, in properly conducted works, should be small. Even when experimental errors of assay be also taken into account, the discrepancy between theoretical

extraction, estimated on assay and tonnage, and actual extraction of fine gold contained in the bullion, should not exceed 3 to 4 per cent. As a matter of fact, chiefly from unskillful work, the bullion actually recovered amounts in the Rand gold fields, on an average, to only 60 per cent; whereas the average extraction, as estimated by assaying the material before and after treatment, amounts to 70 per cent. The chief causes of error in the estimate of extraction arise from incorrectly calculating the tonnage, from faulty sampling of charges and residues, and from careless assaying of too small samples. If truckloads of tailings be weighed periodically and the percentage of moisture carefully tested; if samples of tailings before treatment be taken from each incoming truck and mixed thoroughly, and the residues after treatment be treated likewise; and if assays be daily made in duplicate on not less than an assay-ton of material, when this is low grade—then there should be little difference between estimated and actual extraction, provided there is not much leakage and the gold slimes are carefully handled. In many works a serious loss is incurred by allowing solution, containing very fine gold slimes in suspension, to enter the general stock of solution, and to be ultimately discharged with the residues. So fine is some of this material that it will even pass through the finest filter cloth and remain suspended after hours of 'settling.' The most effectual way of overcoming this difficulty is to run all solution, filtered or decanted, from gold slimes during the process of 'cleaning-up' or separating them from the zinc, into a separate vat, called a 'settler.' This is left undisturbed for some days, after which the supernatant liquid may be safely run off. This settler should equal in capacity the united zinc precipitation boxes, and may be cleaned up half-yearly. Filter presses are sometimes used to remove as much solution as possible from the gold slimes before drying the same." In large works, like those of the Crown Reef Company, the actual returns of bullion amount to 95 per cent of the calculated extraction. It is explained that the difference in the estimation of the weight of tailings treated is sufficient to account for the difference of 5 per cent (G. E. Webber, Jr.). The Nigel Company recover about 93 per cent of the calculated extraction; the incomplete recovery is attributed partly to the "soakage" of the vats, and particularly to the treatment of the bullion, the slag sometimes containing as much as two hundred ounces to the ton, of which only from 60 to 70 per cent are recovered by amalgamation in a grinding-pan (W. A. Radoe).

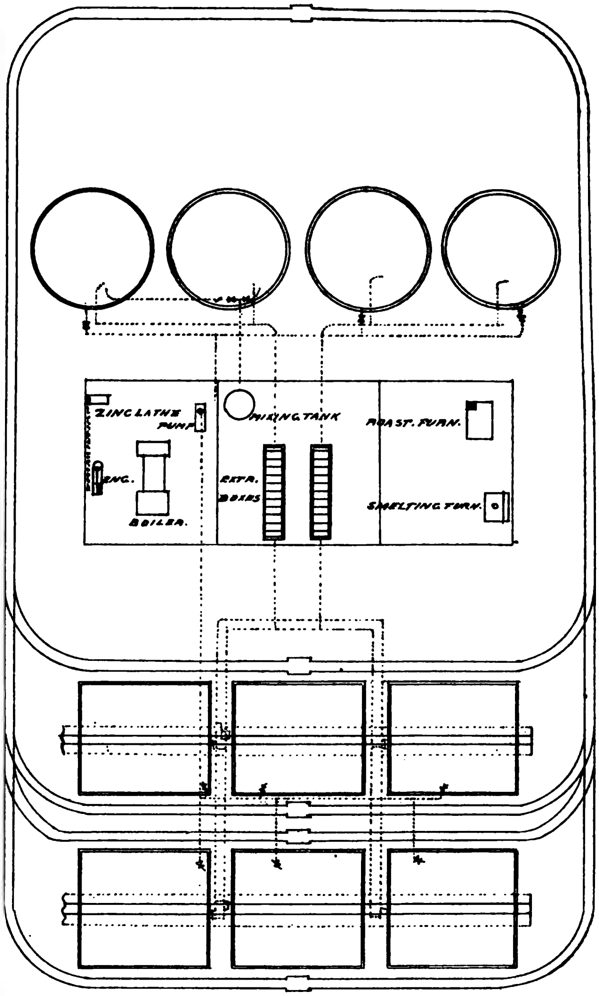
*The Zinc for Bullion Precipitation* "is used in the form of thread-like turnings, obtained as before described. A cubic foot of them weighs from 3 to 6 lbs., and exposes forty square feet of surface per pound weight. Granulated zinc is never used, as it exposes a very small surface in proportion to its weight and is liable to clog in the extractors. Aluminium in conjunction with an electric current has been suggested, as also alternate sheets of iron and lead foil between which a current of two hundred ampères and seven volts passes; but these methods are still in the experimental stage. The consumption of zinc varies greatly in different works, and is dependent upon causes other than the amount of gold precipitated. The absolute amount varies from 2 to 8 oz. per ton of ore, but of this, owing to waste in cutting out and turning, probably not more than one half goes actually into solution, when the finely divided zinc included in the gold slimes be also taken into account. The precipitation of the bullion is conducted as described before. The

zinc consumption, above the amount required for gold precipitation by the equation of the chemical reaction, is due to its solution in the caustic alkali formed as indicated, in the free cyanide and caustic alkali present in the solution as it issues from the leaching vats, and also in its precipitating action upon other substances in solution. An average ore would probably consume about ten times as much zinc as it yielded bullion, but if in treatment caustic soda has been used in excess, the consumption will necessarily be higher."

*Treatment of Precipitates*—"Acid is occasionally used for making a complete clean-up of all the zinc contained in the boxes, and also for refining the amalgam (zinc, gold, and mercury) formed therein, when the tailings have contained much 'floured' quicksilver. Its use, for refining generally, is not advocated in Johannesburg, as it involves washing and filtration of the slimes, and loss of gold by the formation of regulus in melting, if sulphates have remained in the slimes by fault of imperfect washing. The method most in use for refining gold slimes in the South African gold fields is by the use of nitre. The slimes are dried till just before they become dusty; they are then mixed with powdered nitre, the amount varying from 3 to 33 per cent of their weight, and gently heated as a thin layer, either in a wrought-iron pipe (10 in. diameter by 6 ft. length), or preferably in a tray of wrought-iron (⅜ in. thick, by 6 ft. by 3 ft. by 1 ft.), which may also be used for the drying process. In neither case do the flames come into direct contact with the slimes; a hood carries off the obnoxious fumes. By the use of nitre everything in the zinc precipitation boxes which passes a sieve of three of four meshes to the lineal inch may be refined, and thus the finely divided zinc, which otherwise accumulates and clogs in the boxes, is constantly removed. Less nitre is always used than is required to oxidize all the base metals present, as otherwise the free nitre will rapidly corrode the plumbago crucibles, which subsequently are used for melting; it is advisable, however, to remain as near as possible below the limit, as the roasting which follows is thereby conducted quicker and at a lower heat. Besides rendering the bullion finer—containing say only 15 per cent base metals—this nitre-roasting gives a cleaner slag and lessens by at least one half the time required for fusing the gold slimes, and prevents violent ebullitions of vapor from the crucible. From 3,000 to 4,000 oz. of bullion can be obtained in twenty-four hours from roasted slimes containing 33 per cent of gold by the use of No. 70 plumbago crucibles, with good coke, in four box furnaces (20 in. square by 22 in. deep). The following fluxes have been found to answer well: When much metallic oxide is present—slimes six parts, borax four parts, soda two parts, sand one part. When little metallic oxide is present—slimes three parts, borax one part, soda two parts, sand one part. The function of the sand is to form a fusible slag with the soda, and also to protect the pots against metallic oxides and the potash formed by the reduction of the nitre. The slag resulting from melting slimes usually contains an appreciable quantity of gold. This, in the absence of smelting works, is generally crushed by hand in a mortar or by power in a smallest size Gates or Fraser & Chalmer's sample grinder. It is then panned, and the tailings resulting, still rich as a rule, are shipped to Swansea. In estimating the cost of a flux, it should be remembered that a very small percentage of gold in the slag will pay



SECTIONAL ELEVATION




GROUND PLAN

VARIATION No 1.

IN

DESIGNS OF CYANIDE PLANTS

COP. FROM W.A. FELDTMANN'S "NOTES ON GOLD EXTRACTION" ETC.

20 15 10 5 0 10 20 30 40

for an improved flux, and that flux which gives the cleanest, most fluid slag is preferable."

I have given here the mode of bullion treatment in Johannesburg as described by the African Gold Recovery Company; in addition to it, I refer to my own way of procedure, as given on page 36, which may offer some points of advantage.

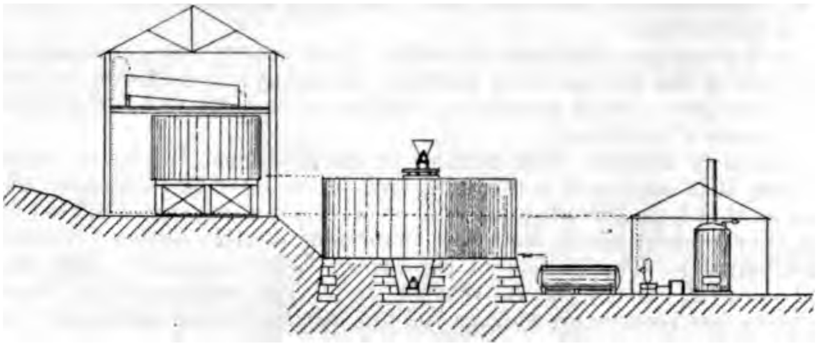
*Fineness of Bullion.*—The bullion in the Robinson Works is about 650 fine; it is very hard and brittle, and the bars are by no means uniform, so that it is difficult to obtain an accurate assay; in addition to zinc, they contain silver, lead, and sometimes a little copper (Butters and Clennell). The bullion of the Crown Reef Company is 950 fine (830 gold, 120 silver) (G. E. Webber, Jr.). The bullion of the Nigel Company has been on an average 795 fine during the last seven months (W. A. Radoe).

*The Cost of Treatment* "varies according to the size of plant and the facilities for working, but exclusive of royalty may be taken at from $1 50 per ton for a 5,000-ton plant, to 87½ cents per ton for plants treating 20,000 tons monthly."

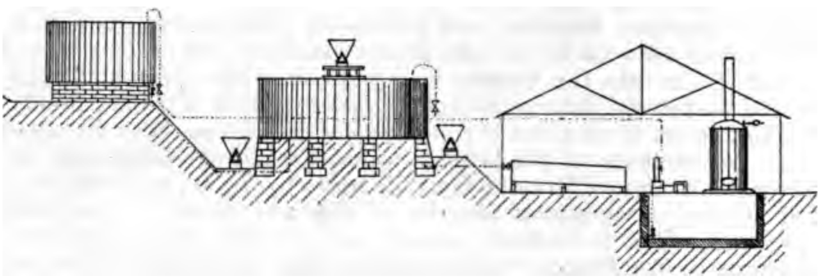
*The Cost of Plants* "varies according to the locality and the style in which they are erected. To erect an average size plant costs at Johannesburg about $6 25 per ton of ore it is intended to treat monthly; for very large plants the cost would perhaps be $5 per ton." All plants have the following main features: Leaching or percolation vats, zinc boxes for bullion precipitation, sumps or tanks for storing solutions, pumps for assisting filtration, and pumps for transporting the liquids. The difference between the various plants consists in the size, form, and material of the vats, the system of charging and discharging the tailings, and the general arrangement of the different parts of the machinery. The construction of vats and the handling of tailings has been discussed above. In reference to the latter, I attach details of discharging appliances, taken from "Notes on Gold Extraction," by W. R. Feldtmann.

*The General Arrangement* may be of different kinds. "The most convenient method is to have solution vats, leaching vats, extractors, and sumps in four tiers, so that each series may be completely drained into that next below it. By this means sufficient solution can be stored in the solution vats, and sufficient room left in the sumps to enable work to proceed for from twelve hours to twenty-four hours without pumping. Many plants, however, have the solution vats and sumps on the same level as the leaching vats; in this case the solution issuing from the last mentioned vats is run through precipitation boxes into a small tank and is continually pumped back when required." I attach plates, which illustrate the variations in the general design of plants with regard to the relative position of the different parts, which with their explanation are taken from "Notes on Gold Extraction by means of Cyanide of Potassium, as carried out on the Witwatersrand Gold Fields," by W. R. Feldtmann.

"In No. 1 design, the leaching vats are placed highest. The solution gravitates from these through the zinc boxes into the storage vats, there to be made up to strength ready for pumping up to the leaching vats again. In the sketch the discharging of the tanks is assumed to be done over the side. In No. 2 design the solution is either pumped direct from the leaching vat, or, running into a small sump or an air-tight receiver, is pumped from there into zinc boxes, and runs thence into

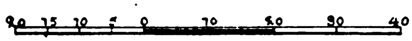


— VARIATION No. 2 —
— IN —
— DESIGNS OF CYANIDE PLANTS. —

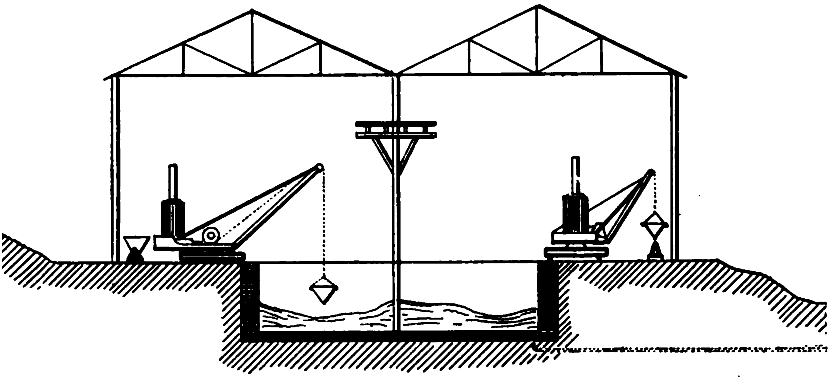


— VARIATION No. 3 —
— IN —
— DESIGNS OF CYANIDE PLANTS. —
— SCALE 16 FT.—1 INCH. —

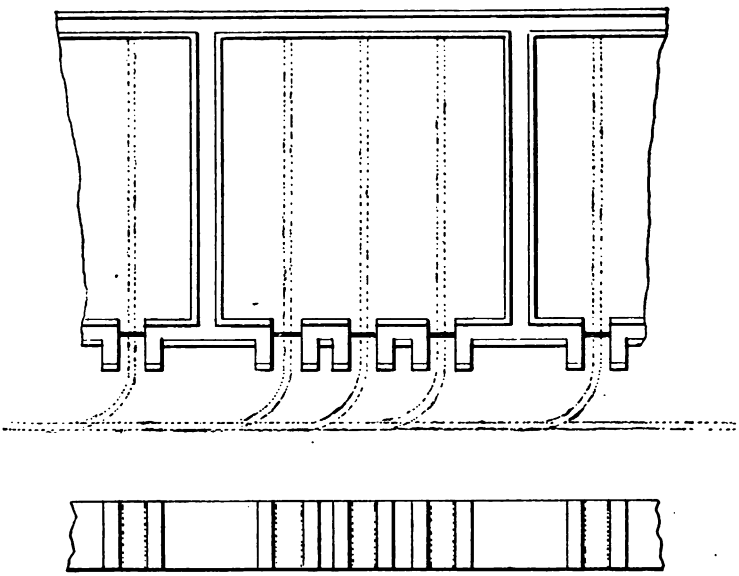
COP. FROM W. A. FELDTMANNS, NOTES
ON
GOLD EXTRACTION" ETC.



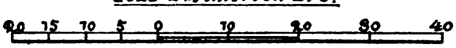

overhead storage vats. Having been made up to strength, it is rea( to run direct into the leaching vats again. The discharge system in( cated is the 'bottom discharge.' No. 3 design is a combination of t two previous ones, and is advantageously fitted with a pipe service enable one, if desired, to run solutions up through the sand in t leaching vats. As shown in the sketch, the plant is designed for si discharge; but of course any system of discharge may be applied any of the three arrangements of plant.



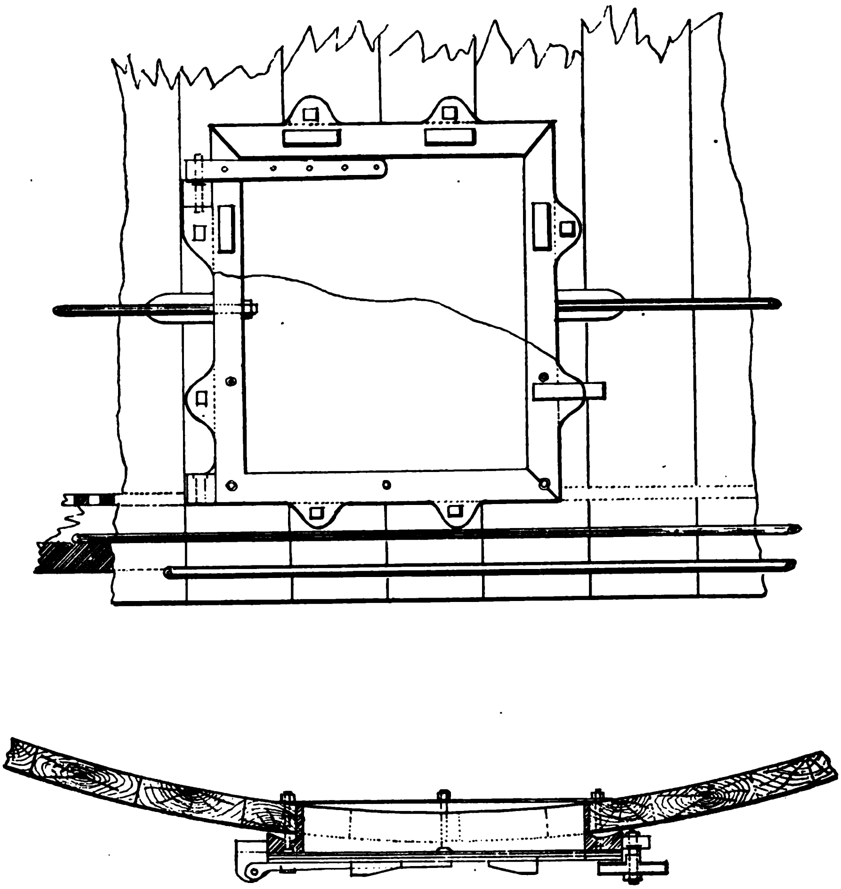
— DISCHARGING TAILINGS VATS —
— AT THE —
— LANGLAAGTE ESTATE Co's PLANT —
— SCALE 20 FEET = 1 INCH —



— SQUARE FILTER VATS AT THE WORKS OF —
— THE CROWN Co —
— WITH DOORS FOR THE DISCHARGING TRUCKS. —
— SCALE 20 FEET = 1 INCH. —
— COP. FROM W.R. FELDTMANN'S NOTES —
ON "GOLD EXTRACTION" ETC.



*Buildings.*—"The majority of plants now erected have only the zinc boxes inclosed in buildings, and there is little objection to having no weather protection when cemented vats only are used, but with wooden vats, exposure to the sun and weather undoubtedly causes increased leakage.

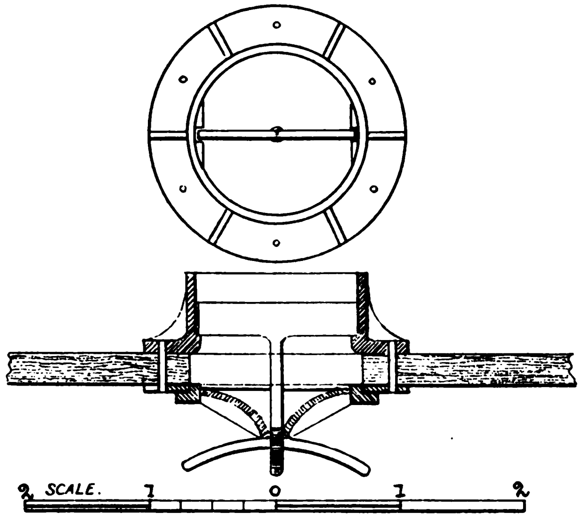


—W. R. FELDTMANN'S—

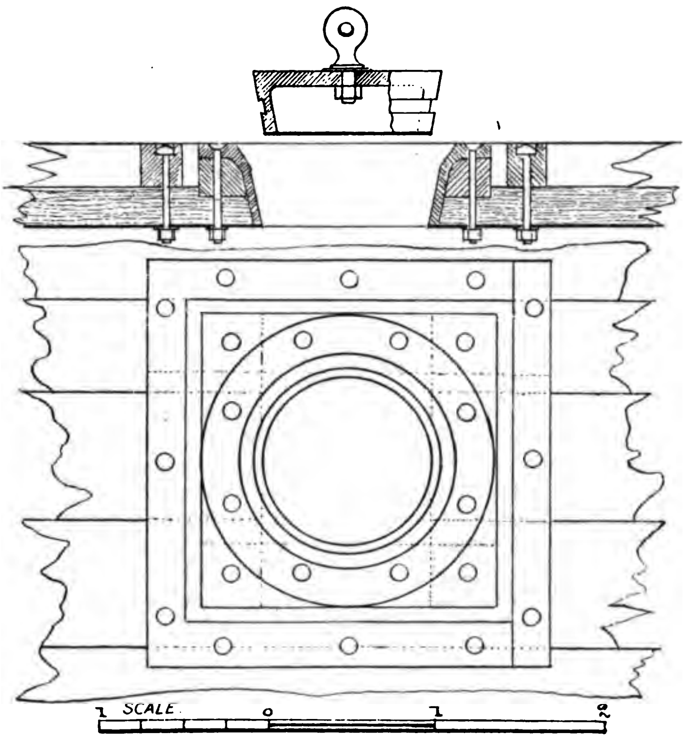
—SIDE DISCHARGE AT PERCOLATION VATS—

—SCALE 1 INCH = 1 FOOT—

*Labor.*—"The men employed in a plant of average capacity (5,000 tons per month) are: One manager, one assayer, two shift men, one mechanic, two native gangers, and the native crew." The Nigel Company employ two white men of twelve-hour shifts in the works, whose duty it is to "make-up," pump in, and drain off solutions, and to attend




*CH. BUTTERS' BOTTOM DISCHARGE AT PERCOLATION VATS.*




*W. E. IRVINE'S BOTTOM DISCHARGE AT PERCOLATION VATS.*

COP. FROM W. R. FELTMANN'S "Notes on Gold Extraction" Etc.

to things generally, and one white overseer over the natives (about thirty in number), who attends to charging and discharging the vats (W. A. Radoe). The Crown Reef Company employ one white man per shift of eight hours and seventy native laborers, handling 500 tons of tailings (G. E. Webber, Jr.). During 1893 about 500 vats were in use on the South African gold fields, of a total daily capacity of 11,000 tons.

*Results of the Process.*—The process of treating tailings by cyanide was first introduced on the Transvaal gold fields in 1890; the output of gold by its means has since taken on enormous proportions. The gold produced by cyanide in 1890 amounted to 286 oz.; in 1891 to 34,862 oz; in 1892 to 178,688 oz.; in 1893 to 330,510 oz.; and during the first six months of 1894 to 317,950 oz. I inclose a table giving the output of gold in the Witwatersrand district by the milling and cyanide process. The value of cyanide gold is on an average $15 per oz. The total value produced by cyanide at the end of June amounted to $12,934,440. This enormous amount has been almost entirely derived from low-grade tailings, which were, before the introduction of MacArthur-Forrest's methods, practically valueless, for want of cheap and efficient means of extraction. The working cost of treatment by the process is the lowest known in the history of gold metallurgy, roasting never, and preliminary treatment in exceptional instances, being required. The process is adopted, only amongst others, by the following mines:

Robinson Company, 60 stamps; in 1892, 101,061 tons were crushed, yielding 98,799 oz. of gold, an average of 19 dwts. 13¼ grs. per ton; the concentrates yielded 8,407 oz., and 75,375 tons of tailings by cyanide treatment 27,577 oz.; the average from all sources was 1 oz. 6 dwts. 16 grs. per ton; the cost per ton about $5 50.

The Langlaagte Estate Company, with 120 stamps working, crushed in 1892, 197,201 tons, yielding on an average 6 dwts. 23.81 grs. per ton; twelve thousand tons of tailings are being worked per month at a working cost of $1 per ton; this plant is being increased to treat thirty thousand tons monthly; the total value of gold received, including that from treatment of tailings by cyanide, amounted in 1892 to $1,554,850. The Langlaagte Estate Company has been using cement tanks with good results for some time past; the vats and sumps are built in excavations made in solid ground, the upper part being level with the ground. A tram-line runs over the center of five large vats, each holding 450 tons, for charging purposes. A steam crane, running on rails, placed on each side of these vats, lifts and empties the tailings, after treatment, into a movable chute placed over two parallel lines of trucks. This chute fills two trucks at each lift at a cost of less than 2 cents per ton, including wear and tear and maintenance of the machinery. The time necessary to empty each vat is fourteen hours.

The following description of the application of the cyanide process in the very extensive works of the Langlaagte Estate and Gold Mining Company in Johannesburg is derived from notes written by the company's assayer, Mr. Thomas Lockhart, which have been contributed by the company's manager, Mr. James Ferguson, whilst this paper was in press:

The ore is of a silicious nature, containing about 2 per cent of iron pyrites, and very little base matter; it is not treated directly by cyanide, but is first put through the battery-amalgamation process. Eight hundred tons are crushed daily. The tailings leaving the plates are con-

centrated, and then run to three settling dams, each of which holds 7,000 tons. The slimes are here separated from the tailings and allowed to run away, as they are greatly impeding the percolation of the cyanide solution in the vats. The tailings, free from slimes, are hauled from these dams in trucks, by means of two endless steel wire ropes, and run onto an overhead tram-line, from where they are dumped directly into the vats for treatment. The vats are built in excavations; their tops are level with the surface of the ground. Of these vats there are fifteen; they are built of brick, and circular in form, of a diameter of 40 ft., and a depth of 9 ft. 3 in. Their sides are cemented, and the bottoms laid with concrete; the latter slope gently to one side, where a 2 in. pipe carries off the solutions. The bottoms are covered with a filter-bed, built of stones, about 3 in. deep, which are packed close together and form a level surface, which is covered with cocoa matting. Each vat holds about 450 tons of tailings, which remain in them under treatment for three days. After the completion of the extraction, the residues are discharged by means of trucks, which are lowered into the vats, filled by Kaffirs, hoisted up by a crane, and run out on the dump by means of an endless rope.

As a rule three vats are charged per day and three discharged. During March 33,000 tons of tailings were treated on 31 working days, or 1,064 tons per twenty-four hours. These tailings averaged before treatment 4 dwts. 5 grs. of gold per ton; the residues, after treatment, contained 17 grs., which corresponds with an extraction of 83 per cent, but only 68 per cent of the assay gold-value was actually recovered. The chief causes of the incomplete recovery of the gold are: The imperfect precipitation by zinc; the presence of slimes in the tailings, which take up and retain some of the cyanide gold solution; the loss of gold solution through leakages in tanks and pipes; and the mechanical losses in manipulating the precipitates in drying, calcining, and melting.

The strength of the first or strong cyanide solution pumped onto the tailings, varies from 0.32 to 0.38 per cent: it stands 3 or 4 in. above the surface of the charge, and remains in contact for from eight to twelve hours; after that time it is run off and passes through the zinc precipitation boxes; it is then brought up to its original strength by adding cyanide of potassium and used again on a next charge. The zinc boxes are 18 ft. long by 5 ft. wide and 2½ ft. deep, and contain the usual divisions. Five such boxes take the solution from ten vats.

The first part of the solution, when running off the charge, contains about 0.02 per cent cyanide; the following portions are of greater strength and rise up to 0.3 per cent. When the solution has drained off, the second, or weak, solution, of about 0.2 per cent cyanide, is pumped on; this percolates directly through and replaces the first solution, which has remained in the material. A water-wash completes the treatment. This wash remains in contact during some hours, and is then drained off as usual; it forms the weak or second solution for a following charge.

The solutions are stored in vats 56 ft. in diameter and 9 ft. 3 in. in depth. The consumption of cyanide amounted during the last six months of 1893 to 0.55 lb. per ton of tailings.

The treatment of the concentrates (sulphurets) is conducted on the same plan as that of the tailings. Two tanks, each 35 ft. in diameter and 2½ ft. deep, serve the purpose. A solution of 0.6 per cent cyanide is used; this remains in contact with the material for six hours; it is

then run off and the gold precipitated as usual in the zinc boxes, which are 11 ft. long, 2 ft. deep, and 2 ft. wide. The solution is then restored to the required strength and used over and over again till the assays of the residues prove the termination of the extraction.

During the month of March 405 tons of concentrates were treated; they averaged before treatment 2 ozs. 10 dwts. 4 grs. of gold per ton; the residues contained 5 dwts. 14 grs., which corresponds with an extraction of 89 per cent of the assay-value; this percentage is actually recovered. The consumption of cyanide per ton of concentrates amounted during the last half of the year 1893 to 0.5 lb. The cyanide used contains 98 per cent of potassium cyanide. The zinc used for precipitating the gold contains 1½ per cent impurities; it is applied in filiform. For every one ounce of fine gold obtained 1.48 lbs. of zinc are consumed. Cleaning up takes place twice a month. The gold precipitates, mixed with finely divided zinc, are separated from the coarse zinc by shaking; they are then dried and calcined. The calcination is carried on on a heavy iron plate, which is heated to redness; it is continued till the oxidation of the zinc is complete. The calcined mass is then fused with borax, soda-ash, and sand. The mixture is charged into No. 50 plumbago crucibles and melted in a reverberatory hearth furnace, which holds twenty-two crucibles at a time. The time required for melting varies from one and a half to three hours, according to the character of the material and the temperature of the furnace. The molten mass is poured into iron moulds; the bullion, thus obtained, is remelted into bars of 600 oz. The fineness of the bullion varies considerably; that obtained from tailings runs from 700 to 780; that from concentrates from 750 to 820 fine. The weight of the fluxed precipitates which are reduced per month amounts to about 9,000 lbs. Three shifts, of two white men each, attend to the working of the process. Two twelve-hour shifts, of five Kaffirs each, are constantly employed in cutting zinc shavings. The charging and discharging of the percolation vats is done by contract; 14 white men and 250 Kaffirs are employed for the purpose. The buildings consist of wooden frame structures, covered on all sides with corrugated iron.

The Langlaagte Company pays a royalty for the use of the MacArthur-Forrest patents, amounting to \$1 25 per standard ounce of gold extracted from tailings, and 62½ cents per standard ounce of gold extracted from sulphurets.

The Ferreira Company derived for the half year ending March 31, 1892, a profit of \$22,030 on the treatment of 15,310 tons of tailings, producing 3,495 oz. of gold at a cost of \$2 91 per ton. For the year ending March 31, 1893, the tailings treated by cyanide yielded 11,201 oz. of gold, which means to say 4,592 dwts. of the value of \$4 16 per ton.

The Crown Reef Company has 210 stamps working. For the half year ending March 31, 1893, the tailings treated by the process yielded 16,629 oz. of gold. The revenue from tailings and concentrates amounted to 92 cents per ton.

The Henry Nourse Company treated, during the first six months of 1893, 12,640 tons of tailings by cyanide, yielding 3,040 oz., or 4.81 dwts. of gold, at an average cost of \$2 33 per ton.

The Meyer and Charlton Company treated during six months ending June 30, 1893, 10,799 tons of tailings by the cyanide process, yielding 3,205 oz., or 5.93 dwts. per ton.

The Nigel Company obtained during twelve months 21,471 oz. of gold by the process from tailings and concentrates.

The Rand Central Ore Reduction Company has been formed to purchase tailings from companies who have not erected their own plants; 1,500 tons of tailings are treated in their works per twenty-four hours. The vats, made of wood, hold from 75 to 600 tons. The tailings average 5 dwts. before and 1 dwt. after treatment. The strong solutions contain about 0.3 per cent, the weak about 0.15 per cent of cyanide. The last solution runs off at about 0.08 per cent. All solutions together amount to about 1 ton per ton of ore. The extraction takes about three days. The cyanide used is of 98 per cent on the average. The extraction from the tailings varies from 75 to 80 per cent of the assay-value, of which 80 per cent are actually recovered; the clean-up is never complete and much gold remains in the slag. The zinc consumption amounts to about 1 lb. per ounce of gold recovered. Electricity is also used for bullion precipitation. The bullion after melting is 750 to 850 fine. The total cost of treatment is about $1 per ton. The cost of the plant amounted to $600,000; 200 white men and 800 Kaffirs are employed. (D. Ruston.)

The Langlaagte Royal Gold Mining Company is erecting a cyanide plant for the treatment of 10,000 tons per month. The mentioned companies, in conjunction with a number of others, more or less important, treat upwards of 200,000 tons of tailings monthly. During June, 1894, 220,507 tons were treated, with an average yield of $3 per ton. The treatment of concentrates by chlorination has been abandoned in favor of cyanide treatment by the Crown Reef and the Langlaagte Estate works.

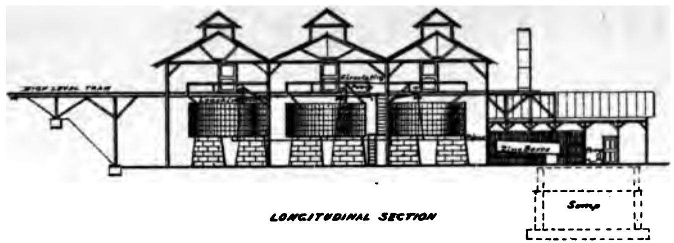
The large profits accruing to some of the mining companies are to a large extent derived from their tailings-treatment. Individual returns show that at several of the leading and most prosperous mines 35 to 50 per cent of their output is due to their using the cyanide process. The Nigel Company's official report for the three months ending September 30, 1893, shows that a net profit of $122,325 was made, of which only $43,970 was derived from amalgamating, and $73,895 from cyanide treatment of tailings and concentrates. The report of the Langlaagte Estate Company for the same period shows a net profit of $51,410, derived from the same source. During 1892, 37,595 oz. were recovered by the process from Robinson tailings, 19,482 oz. from Crown Reef, and nearly 29,000 oz. from Nigel tailings. All companies mentioned use the cyanide process as described in the MacArthur-Forrest patents, and use zinc as bullion precipitant. The amount of gold recovered by the Molloy process (see page 38) is insignificant. The two companies which were described in the official list of the Witwatersrand Chamber of Mines as using that process, are now mentioned as using the MacArthur-Forrest process.

The cyanide plants at Johannesburg are, generally speaking, very similar to each other in their construction, and the description of a typical one, that of the Robinson Company, will sufficiently illustrate their construction. The plant in question, of which the attached plans were designed by Mr. Chas. Butters, consists of twelve circular open leaching vats, each having a capacity of 2,000 cub. ft. and holding 100 tons of tailings and cyanide solution. The vats, built upon elevated arched stone foundations, are filled from a high level tramway above them, and emptied through trap doors in the center into tramcars below.

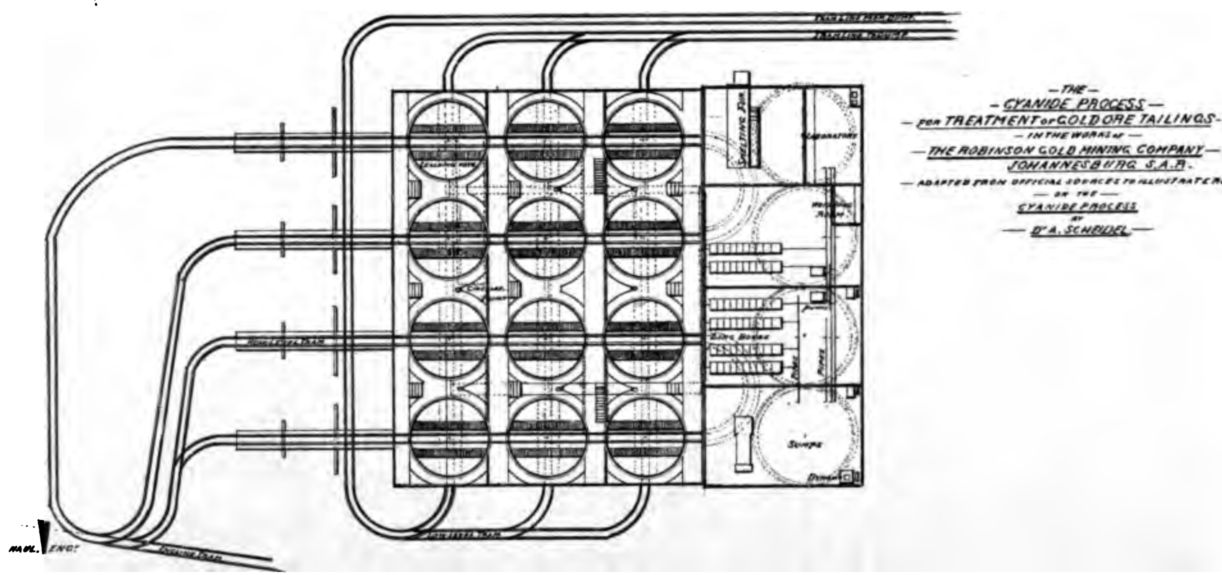
Next in order is a series of precipitating boxes, designed to continuously precipitate gold from the solution as it passes from the leaching vat to the sump. The boxes are 20 ft. long, 2 ft. wide, and 2 ft. deep; they have inclined bottoms. They are divided into compartments a 20 in. in length each. Each chamber contains about 40 lbs. of zinc turnings. Seven compartments in each box are filled with shavings; of single compartment at the head is left empty to receive any sand that may be carried through the filter by the solution from the tanks. A double compartment at the foot is also left empty to allow any gold that may be carried away by the stream of liquid to deposit before the solution flows into the sump. About 60 tons of solution, which is the quantity required for treating the ordinary daily charge of 225 tons of tailings, is allowed to run through two zinc boxes in about nine hours. This solution may carry from 1 oz. to 3 oz. of gold per ton of liquid; after passing through the zinc boxes it rarely contains more than $2, and should not contain more than 50 cents if the precipitation has been properly carried out (Butters and Clennell). Underneath the leaching vats are four 200-ton sumps, brick and cemented tanks set in the ground. In these sumps are prepared and stored the solutions for dissolving and washing out the gold. On the top of them are placed a double set of duplex pumps, so arranged that both, or either, will throw from any sump into any leaching vat. On the tram-level above the vat is a 10 horse-power double-drum winding engine and boiler, employed to hoist the tailings out of the settling pits. The plant includes four furnaces of large capacity for smelting and refining bullion; also a laboratory and weighing-room. An elaborate system of tram-lines is laid down, on both high and low levels, for delivering and discharging the tailings in the most direct and efficient manner. Works, pits, and dumps are lighted by electric light. Lathes for turning zinc are employed, and a 12 horse-power engine, with 16 horse-power boiler, supply all power. The whole is covered by an airy building of wood and iron. The methods of working are: Hoisting tailings and filling vats; pumping cyanide solution onto the tailings in the vats to dissolve the gold; running off this gold solution into zinc boxes and precipitating the gold; return of the cyanide solution into the sump below for repeated use; collecting, melting, and refining the precipitated gold (M. S. P.).

The report of this company for the year ending December 31, 1893, shows that 55,200 tons of tailings were treated by the cyanide process, yielding 17,921 oz. of gold. The cost at the cyanide works is given per ton of tailings treated, as follows: Wages, 29.62 cts.; supplies, 12.24 cts.; fuel, 10.48 cts.; cyanide, 48.38 cts.; zinc, 2.26 cts.; filling and discharging vats, 37.62 cts.; royalty, 16.38 cts.; total cost, 173.86 cts. per ton. The actual cost of treatment per ton, omitting royalty, was 156.98 cts. The average extraction by the process was 68.7 per cent of the assay-value of the tailings. An innovation in percolation in the Robinson Works consists in the circulating system, which has been described by Butters and Clennell as follows:

"It has been stated that in the usual method of working about a ton of solution is employed in the treatment of a ton of ore. Since, with free-milling ore, a much smaller quantity is sufficient to dissolve the same percentage of gold, it was suggested that the solution from one tank might be transferred to a second, and be made to dissolve an additional quantity of gold before being passed through the zinc boxes; for




LONGITUDINAL SECTION




PLAN

— THE —
— CYANIDE PROCESS —
— FOR TREATMENT OF GOLD ORE TAILINGS —
— IN THE WORKS OF —
— THE ROBINSON GOLD MINING COMPANY —
JOHANNESBURG S.A.R.
— ADAPTED FROM OFFICIAL SOURCES TO ILLUSTRATE REPORT —
— ON THE —
CYANIDE PROCESS
BY
— D^R A. SCHEIDEL —

example, it was found at the Robinson Works that 20 tons of solution were amply sufficient to extract 40 oz. of gold from 75 tons of tailings in one tank. It was found that 20 tons of solution sufficed to fill a tank holding the usual charge of 75 tons of tailings, covering the charge to a depth of three or four inches. Instead of replacing these 20 tons of solution by fresh cyanide, the solution filtering through was continually pumped back again into the same tank for about thirty-six hours and then passed through the zinc box. The extraction of gold by this circulation-system was equal to that obtained by the ordinary method, and the consumption of cyanide was much less, since a much smaller quantity of solution was exposed to the action of the zinc. A further modification suggested itself, namely, the transference of the solution charged with gold from one tank to a second and third, in order that it might take up an additional quantity of gold from fresh tailings before passing into the zinc boxes. The advantages of this method are that the solutions from which the precipitate is obtained are much richer in gold, giving a cleaner deposit on the zinc, with much less consumption of cyanide."

In the Durban-Roodeport Company's works the extraction ranged from 67 to 85 per cent. The cost of treating the tailings, including patent-royalty, amounted to $1 54; the profit to $2 87 per ton. During eleven months, in 1893, 79,765 tons of tailings were treated, producing 22,751 oz. of gold, adding about 33 per cent to the total revenue.

The New Chimes Gold Mining Company have commenced to treat their tailings in their own cyanide works only since the beginning of the year; during March, 4,180 tons of tailings realized 709.55 oz. of bullion. The assay-value amounted to (fine gold) 3.33 dwts.; fine gold saved, 2.25 dwts.; extracted, 67.56 per cent; lost in tailings, 32.44 per cent; value of bullion per ton treated, $2 36; expenses, $1 02; profit per ton, $1 32. (G. Halford Smith.)

The financial success of the cyanide process in South Africa is best proved by the dividends paid by the mining companies which use cyanide for tailings treatment. The following is a list of dividends paid during 1893 by companies under such conditions:

| Names of Companies. | Dividends. | Amounting to— |
|---|---|---|
| City and Suburban | 100 per cent | $425,000 |
| Crown Reef | 50 per cent | 300,000 |
| Durban-Roodeport | 45 per cent | 282,250 |
| Ferreira | 100 per cent | 225,000 |
| Langlaagte Estate | 30 per cent | 705,000 |
| Meyer & Charlton | 60 per cent | 215,100 |
| New Primrose | 40 per cent | 391,870 |
| New Rietfontein | 25 per cent | 200,000 |
| Robinson | 8 per cent | 1,087,500 |
| Nigel | 50 per cent | 400,000 |

The total output of the Rand mines for the year ending June 30, 1893, apart from the cyanide process, was 1,087,058 oz.; by the process this quantity was increased by 226,078 oz., making a total of 1,313,136 oz. From districts not included in the Rand proper, a further recovery of 2,395 oz. was returned, making in all 228,473 oz. due to the working of the process. It will thus be seen that, by the use of the process, the Rand production was increased by 21 per cent. At several of the lead-

ing and most prosperous mines, 35 to 50 per cent of their gold output is due to the use of cyanide. The report of the Witwatersrand Chamber of Mines gives the output of that district of the Transvaal, for March of this year, at 165,372 oz., from fifty-three mines and three custom works; of which 44,664 oz., of the value of $668,655, were extracted by cyanide from 204,421 tons of tailings, and 1,367 oz., of the value of $20,805, from concentrates by the same means. The returns per ton of tailings averaged 4.37 dwts.; 33.74 per cent of the total month's production of gold is derived from very low-grade material by the process. The cyanide process is the only one which is successfully treating tailings on a commercial scale. Its economical importance for Johannesburg will be evident from the following tables:

*Output of Gold in the Witwatersrand District by Mills and Cyanide, in ounces.*

| Month. | 1889. | 1890. | | 1891. | | 1892. | | 1893. | | 1894. | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Mills. | Mills. | Cyanide. | Mills. | Cyanide. | Mills. | Cyanide. | Mills. | Cyanide. | Mills. | Cyanide. |
| January | ......... | 35,006 | ......... | 52,595 | 610 | 72,589 | 11,971 | 91,236 | 17,138 | 106,263 | 43,551 |
| February | 22,457 | 36,887 | ......... | 48,532 | 1,547 | 75,752 | 10,897 | 76,889 | 16,363 | 107,829 | 44,041 |
| March | 27,919 | 37,779 | ......... | 51,348 | 1,601 | 81,771 | 11,473 | 91,471 | 20,002 | 115,882 | 49,490 |
| April | 27,028 | 38,696 | ......... | 54,726 | 1,645 | 82,062 | 13,500 | 91,966 | 20,086 | ......... | ......... |
| May | 35,028 | 38,836 | ......... | 53,612 | 1,061 | 86,217 | 13,219 | 92,906 | 24,004 | ......... | ......... |
| June | 30,877 | 37,419 | ......... | 54,263 | 1,600 | 87,752 | 15,500 | 96,708 | 29,199 | ......... | ......... |
| July | 31,091 | 39,226 | 230 | 52,750 | 2,174 | 85,084 | 16,195 | 98,278 | 27,891 | ......... | ......... |
| August | 30,519 | 42,807 | 56 | 55,524 | 3,546 | 83,928 | 16,392 | 101,773 | 34,296 | ......... | ......... |
| September | 34,143 | 45,486 | ......... | 61,993 | 3,609 | 91,148 | 16,702 | 96,322 | 33,463 | ......... | ......... |
| October | 32,214 | 45,248 | ......... | 69,028 | 3,765 | 94,836 | 17,171 | 101,726 | 34,956 | ......... | ......... |
| November | 33,721 | 46,782 | ......... | 68,020 | 5,373 | 89,098 | 17,695 | 101,825 | 36,815 | ......... | ......... |
| December | 39,050 | 50,351 | ......... | 71,981 | 8,331 | 73,184 | 17,973 | 107,060 | 39,297 | ......... | ......... |
| Totals | 344,047 | 494,523 | 286 | 694,372 | 34,862 | 1,005,421 | 178,688 | 1,147,960 | 330,510 | ......... | ......... |

This table is taken from "Notes on Gold Extraction," by W. R. Feldtmann.

*Table showing Companies in the Transvaal Treating Tailings and Concentrates by Cyanide in 1893.*

| Company. | Tons. | Total Output in ounces for 1893. | | |
|---|---|---|---|---|
| | | Plates. | Concentrates. | Tailings. |
| Champ d'Or | 17,896 | 5,722.7 | ........ | 1,787.17 |
| City and Suburban | 49,805 | 37,777.14 | ........ | 9,034.3 |
| Crown Reef | 118,244 | 51,688.0 | 723.2 | 29,679.14 |
| Durban-Roodeport | 78,651 | 37,883.6 | ........ | 22,751.0 |
| Evelyn | ........ | ........ | ........ | 2,545.19 |
| Ferreira | 47,376 | 43,978.1 | ........ | 11,697.19 |
| Geldenhuis Main Reef | 9,495 | 3,417.7 | ........ | 372.17 |
| Gipsy | 3,931 | 1,563.6 | ........ | 712.14 |
| Henry Nourse | 19,749 | 15,329.18 | ........ | 6,893.3 |
| J. H. Burg, Pioneer | 17,606 | 9,073.0 | ........ | 1,120.4 |
| Jubilee | 43,673 | 24,774.10 | 36.0 | 5,254.16 |
| Langlaagte Estate | 222,732 | 65,812.12 | 9,047.11 | 30,050.15 |
| Langlaagte Block B | 64,066 | 19,621.4 | 100.0 | 6,869.19 |
| Marais Reef | 935 | 937.5 | ........ | 419.10 |
| May Consolidated | 60,298 | 24,957.4 | ........ | 2,875.0 |
| Meyer & Charlton | 34,197 | 27,328.12 | ........ | 6,854.2 |
| New Heriot | 21,455 | 14,089.2 | ........ | 8,689.18 |
| New Chimes | 33,641 | 14,510.9 | ........ | 7,296.16 |
| New Primrose | 141,464 | 57,574.8 | ........ | 26,203.18 |
| New Rietfontein Estate | 24,048 | 28,168.12 | 79.18 | 6,957.15 |
| New Spec Bona | 22,289 | 8,784.5 | ........ | 1,040.0 |
| Nigel | 22,273 | 25,455.0 | 3,516.2 | 17,036.8 |
| Orion | 34,657 | 8,318.3 | ........ | 9,677.15 |
| Paarl Central | ........ | ........ | ........ | 957.15 |
| Randfontein | 54,652 | 23,310.16 | ........ | 6,623.14 |
| Robinson | 94,842 | 104,222.17 | 10,659.18 | 17,921.4 |
| Salisbury | 24,786 | 19,268.18 | ........ | 5,587.6 |
| Simmer & Jack | 103,798 | 38,904.12 | ........ | 767.0 |
| Stanhope | 22,858 | 10,790.8 | ........ | 3,873.16 |
| Treasury | 12,429 | 7,587.12 | ........ | 4,284.16 |
| Village Main Reef | 11,607 | 6,143.17 | ........ | 1,996.8 |
| Vulcan | 2,766 | 764.5 | ........ | 50.0 |
| Wemmer | 27,654 | 22,705.13 | ........ | 3,063.8 |
| Witwatersrand | 34,081 | 12,441.13 | ........ | 7,882.18 |
| Customs Works | ........ | ........ | 38,574.0 | 35,669.2 |

*Witwatersrand Customs Works, 1893.*

| African Gold Recovery Company. | | Rand Central Ore Red. Company. | | Robinson Company. | | Total. | |
|---|---|---|---|---|---|---|---|
| Concentrates. | Tailings. | Concentrates. | Tailings. | Concentrates. | Tailings. | Concentrates. | Tailings. |
| 463.18 | 4,130.6 | 9,774.7 | 31,538.16 | 28,335.15 | ........ | 38,574.0 | 35,669.2 |

Gold is valued at $17 50 for plate gold, and $15 for cyanide gold. (These figures are taken from M. I., vol. 2.)

The table giving the monthly analysis of gold production in the Witwatersrand district, for April, 1894 (see Appendix), which has been published by the Witwatersrand Chamber of Mines, will further illustrate the importance of the cyanide process in that district.

**Cape Colony.**—In the British Colony at the Cape "the gold mining industry has not developed to such proportions as to lead to the introduction of the cyanide process." (Letter of Secretary of Agriculture, 27th April, 1894.)

## B. Australasia.

(a) **New Zealand.**—A very successful field for the cyanide process has been the eminently progressive British Colony of New Zealand, where various classes of ore, tailings and concentrates, of a very refractory type, have been and are being treated on a large and commercially successful scale. The colony contains the largest cyanide plant outside of South Africa, that of the Waihi Company, with thirteen vats, where ore is treated at the rate of 2,000 tons and tailings at the same rate per month. The Crown mine at Karangahake is equipped with a smaller but equally efficient plant for ore treatment. Smaller plants for treating ores and tailings are distributed over the Hauraki gold fields. An extensive and very successful agitation plant for the treatment of concentrates is connected with the reduction works of the Sylvia Company, Tararu, Thames. The first mine to adopt the process has been the Crown mine in the Upper Thames District. The first plant was erected in an almost inaccessible position in 1889, under conditions which precluded a success. New works have since been erected by Mr. MacConnell, which are in full and successful operation. The ore is clean quartz, with no sulphurets of base metals; the free gold is very finely divided. The silver is in form of sulphide; some of the gold in form of a telluride. The works are described in the New Zealand Government Mining Report of 1893, by Mr. H. A. Gordon, the Government inspecting engineer, as follows:

"The ore, when brought into the works, is first dumped onto a grizzly; what will not go through the bars runs down to the rock-breaker and is broken up to a maximum size of 2 in. diameter, and then falls into the same hopper where the fine material went. The ore passes then into the drying kilns, which are built of brick, the hot air being confined in a long flue, having a series of steps to prevent the ore from traveling down too fast before it gets thoroughly dried. There is a cast-iron plate at the bottom of this flue, which can be turned to allow of the dried ore to pass down into a large hopper, made of steel plates, $\frac{5}{16}$ in. thick, from which the Challenge ore-feeders are fed. These kilns are only for drying the ore, and not in any way to calcine it. There are two of these kilns built on a stone foundation and placed about 6 ft. apart, the foundation going all the way across. The kilns themselves stand about 30 ft. in height, the step-flue being at an angle of about 30° to 40° from the vertical. There is a furnace at the bottom, where either coal or firewood can be used to dry the ore.

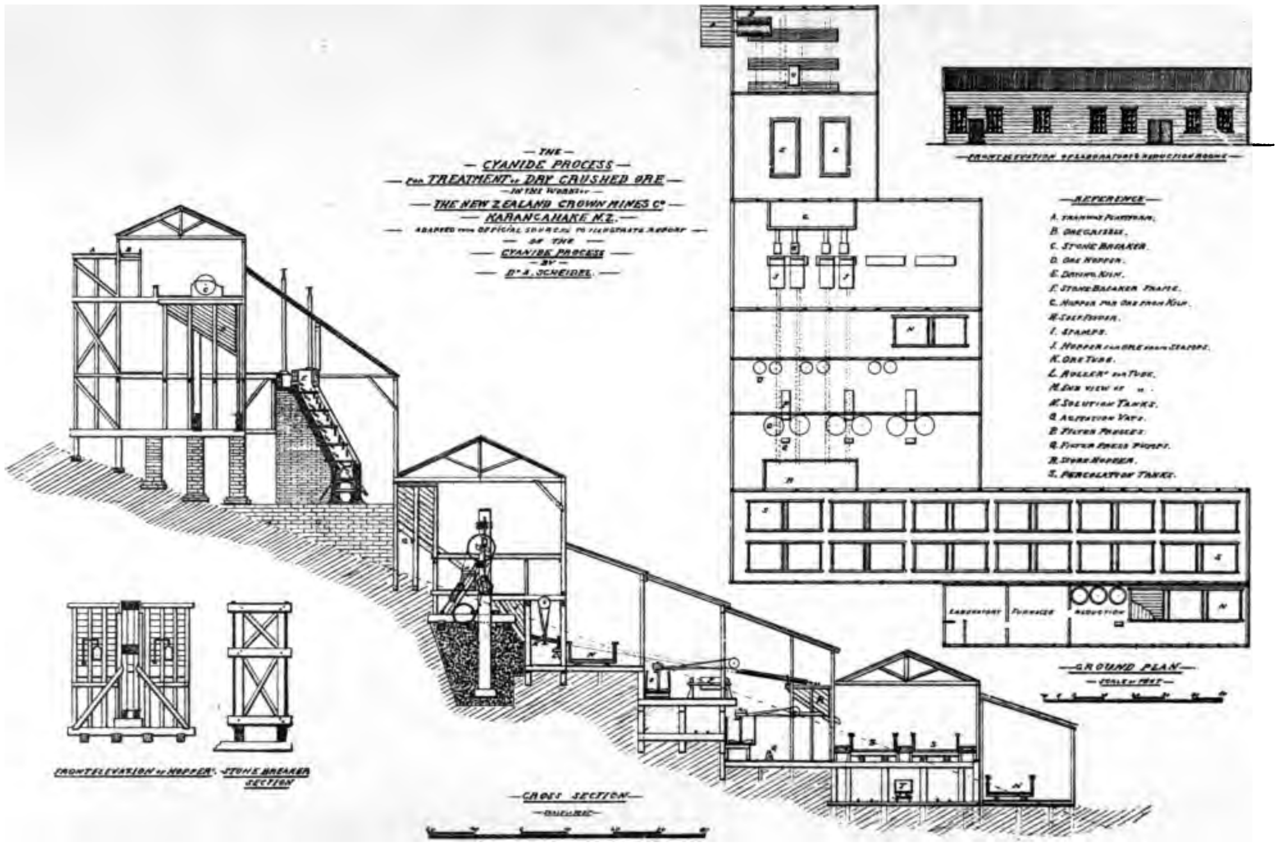
"*Stamp Mortars.*—There is first a concrete foundation put in for the stamps, and on the top of the concrete the stamp mortars are placed on the end of a log of kauri, each 18 ft. in length, 4 ft. 8 in. one way, and 2 ft. 2 in. the other. These are firmly embedded in the concrete, and all bolted together, so as to form a solid block of timber standing on end, having a length of 18 ft. 8 in. by a width of 2 ft. 2 in., and on this the four mortars are placed. They are fitted with screens, having the top standing outward at a slight angle, and held to the face of the mortars by means of a long wedge, the gratings being 30-mesh, equal to 900 holes to the square inch.

"*Stamps.*—The stamps are fitted with the latest appliances for raising and holding them up, the cams and tappets being all constructed on the American type. They are intended to make about ninety-two blows

per minute, having a drop of six inches. The guides and framing are made of wood. Each ten-head battery is driven by a separate belt, and there is further provision made so that twenty additional stamps can be erected should they at any time be required. The pulverized material from the stamps falls into a chute and is conveyed into another set of hoppers at a lower level than the stamp mortars, and from these hoppers the pulverized dust is taken to the leaching vats.

"*Cyanide Plant.*—This consists of twenty-four wooden vats, each 11 ft. long by 9 ft. wide, and 3 ft. 9 in. deep. In the bottom of these vats there is a false bottom, or grating, placed about 3 in. above the ordinary bottom, and on this false bottom a filter-bed is placed, about 4 in. in thickness, the bottom layer being of coarse quartz-gravel and gradually getting finer up to the top, the last coating being fine sand, having a coarse cloth placed over the top of the filter-bed to prevent the sand from being disturbed, as the vats get cleaned out after every charge of pulverized ore. There are also 14 agitators, 8 of which are 5 ft. deep by 4 ft. 9 in. in diameter, and 6 of them 6 ft. deep and 5 ft. 6 in. in diameter. The agitators and vats are all made of kauri timber, the staves of the agitators being 3 in. in thickness, and the vats being made of partly 3 in. and partly 4 in. timber, and all bolted together. Into each of these vats are placed three pipes, under the false bottom, so that the first, second, and third solutions can be drawn off into separate channels. On one side of each vat there is a door, which can be opened to admit of the material being sluiced out after the whole of the cyanide solution is completely washed out of the ore, the solution passing through a long series of boxes filled with zinc shavings, which precipitate both the gold and silver in the form of a blackish powder. There are also three concrete sumps, each 15 ft. by 12 ft., and 6 ft. deep, capable of holding about 30 tons of the cyanide solution; this is pumped up to the vats on the floor above as required. It is in these concrete sumps where the solution is always made up to the proper strength before being used. It is also proposed to use a vacuum pump to assist the filtration of the solution through the pulverized material in the vats. Annexed are plans of the company's plant, to which the following description or reference applies: At point A, the ore is delivered at the battery and tipped onto grizzly, B; the 'fines' pass through and are conveyed to hopper, D; the 'roughs' pass over the grizzly onto the stone-breaker floor and are passed through stone-breaker, C, and fall into the hopper underneath, marked D; the drying-kiln, E, is charged from this hopper. The ore, after passing through the kiln, being perfectly dry, is run into an iron hopper, G, from where it is automatically fed into stamps, I, by self-feeders, H; the ore, after passing through the stamps, is received in hoppers, J, and then conveyed by means of revolving tube, K, either into trucks for conveying ore to agitation-cylinders for treatment, or, if the ore can be better treated by percolation, to store-hopper, R, in connection with percolation plant, from where it is trucked along the top of and tipped into percolation tanks, S, for treatment. The plant is so arranged that the ore, after it is delivered above the stone-breaker, passes from stage to stage by gravitation, requiring the least possible handling, and thereby reducing the cost of labor to a minimum.

"*Crushing Machinery.*—One Lamberton stone-breaker, capable of reducing 70 tons of ore per day fine enough to feed into stamps; and


— THE —
— CYANIDE PROCESS —
— FOR TREATMENT OF DRY CRUSHED ORE —
— IN THE WORKS OF —
— THE NEW ZEALAND CROWN MINES Co —
— KARANGAHAKE N.Z. —
— ADAPTED FROM OFFICIAL SOURCES TO ILLUSTRATE REPORT —
— OF THE —
— CYANIDE PROCESS —
— BY —
— Dr A. SCHEIDEL —
FRONT ELEVATION OF LABORATORY & REDUCTION ROOMS
— REFERENCE —
A. TRAMWAY PLATFORM.
B. ORE GRIZZLEY.
C. STONE BREAKER.
D. ORE HOPPER.
E. DRYING KILN.
F. STONE BREAKER FRAME.
G. HOPPER FOR ORE FROM KILN.
H. SELF FEEDER.
I. STAMPS.
J. HOPPER FOR ORE FROM STAMPS.
K. ORE TUBS.
L. ROLLER FOR TUBS.
M. END VIEW OF "
N. SOLUTION TANKS.
O. AGITATION VATS.
P. FILTER PRESSES.
Q. FILTER PRESS PUMPS.
R. STORE HOPPER.
S. PERCOLATION TANKS.
LABORATORY
FURNACES
REDUCTION
— GROUND PLAN —
— CROSS SECTION —
FRONT ELEVATION OF HOPPER.
STONE BREAKER SECTION

20 heads of 9 cwt. dry stamps, crushing 30 tons of ore per day through a 30-mesh screen.

"*The Percolation Plant* consists of 24 tanks, capable of holding each a charge of 7 tons of finely pulverized ore. The bottom of each tank is covered with a sand and gravel filter. The ore is trucked into the tanks from the storage hoppers. A dilute solution of cyanide is then run on the top, and allowed or assisted to percolate through the body of the ore. As the solution percolates, it is carried away from underneath the filters by means of iron pipes, and permitted to run through a series of boxes of zinc turnings.

"*Agitation Plant.*—This consists of sixteen wooden tubs, fitted with revolving paddles, in which the ore and cyanide solution are agitated together until the gold and silver are dissolved. The pulp is then filtered by means of filter-presses, and the bullion deposited from the solution on the zinc, as already described. The extraction of bullion is given as 93 per cent of the gold and 79 per cent of the silver assay-value. The cost of treatment is $3 50 per ton." This is the only company in New Zealand which does not pay any royalty to the owners of the MacArthur-Forrest patents, the patentees owning part of the mine. The total bullion-value produced by the cyanide process in these works amounts to upward of $142,000.

Another mine of importance, the Waihi Company, has recently adopted cyanide treatment for their ores, supplanting unsatisfactory pan-amalgamation. The ore of that mine is very similar to that of the Crown mines. The bullion recovered by amalgamation has never exceeded 66 per cent of the gold and 40 per cent of the silver assay-value. Experiments on a large scale, made nine months ago, led to the construction of an extensive percolation plant, by which upwards of 20,000 tons of ore have been already successfully treated. The extraction varies from 89 to 91.8 per cent of the gold, the silver extraction from 46.5 to 51 per cent of assay-value. The cost per ton for cyanide and zinc is $1 37½. The gold returns from cyanide treatment are 25 per cent higher than from pan-amalgamation. The ore, which required 60-mesh screens for amalgamation, is sufficiently fine for cyanide if passed through 40-mesh, which means an increased output from the mill of at least 25 per cent, the running expenses remaining virtually the same. Eventually 30-mesh wire gauze may be used. The strength of solution used is from 0.25 to 0.4 per cent. The percolation and subsequent washings can be done in four days. No difficulties have been found in percolation, as the dry ore does not form slimes as wet ore probably would. After the first percolation is finished, the subsequent washings are hastened by atmospheric pressure by means of a vacuum pump. The extra profit by cyanide treatment of Waihi ores over pan-amalgamation amounts to about $3 75 per ton. The company has been experimenting with the Otis crusher, as a substitute for the dry-crushing stamp battery; the results have been unsatisfactory. A royalty of 7½ per cent on the bullion produced is paid to the owners of the MacArthur-Forrest patents, the Cassel Company of Glasgow. For the information in reference to the Waihi Company's cyanide operations, I am indebted to the company's manager, Mr. R. Rose.

A fuller description of the working of the process has since been given by Mr. Barry in the report of Mr. H. A. Gordon, the Inspecting Engi-

neer of Mines to the New Zealand Government (New Zealand Mining Report, 1894), as follows:

"The ore is first dried in open kilns, excavated in tufaceous sandstone. These are 37 ft. deep by 20 ft. in diameter at the top, and taper down to the bottom, where they are finished off with a brick arch, having a door and an iron chute for discharging the dried ore into trucks. These kilns are first charged with wood and ore in layers, each layer of wood being about 5 ft. apart. After the kiln is fully charged, the wood is lighted, and after being all burned up, about one half of the charge is withdrawn—50 tons—and another 50 tons of raw ore, together with wood, added on the top; after which about 50 tons is withdrawn every third day. This method of drying the ore is found to be very economical as regards fuel, as there is not a large surface of cold material to heat up, as is the case with smaller kilns, which are emptied at each charge. The cost of firewood used in large kilns is 37½ cents per ton of ore dried. After the ore is taken from the kilns, it is then put through the rock-breaker, from which it falls into a hopper, and thence, by automatic feeders, it is fed into the stamp-mortars, when it is pulverized until it passes through a 30-mesh and sometimes a 60-mesh screen. It is intended in the future to use a 40-mesh standard. As the pulverized dust passes through the screens it falls into a narrow trough, when it is conveyed by means of an Archimedian screw into a dust-bin at one end of the battery, and from this bin the pulverized material is lifted with a bucket-belt elevator and discharged onto an 8 in. rubber belt with rope edges, and conveyed to and across the hopper 110 ft. long, running the entire length of the cyanide plant-house. This hopper has twenty doors for discharging the sand into the trucks, which are then run straight out over the percolating vats into travelers, running on rails, which are fitted with hand-traversing gearing, enabling a truck to be tipped at any part of the vat. This is an important point, as sand has a tendency to pack if moved about or touched in any way after being tipped into the vat. As a further preventive against packing, there is a small traveler running under the main traveler, with a platform just at the height that the sand is to be filled up to. All trucks are tipped over this platform, which breaks the fall and throws the sand off in a light shower all round. When the vats are filled up to a depth of about 2 ft., a strong solution of cyanide—0.4 per cent—is introduced into the bottom of the vat under the filter-cloth, and forced up through the sand until it stands about 2 in. above it. The solution remaining under the filter-cloth is then drawn off, and filtration commences; the 2 in. on the surface taking about twenty-four hours to percolate through. After the whole of the strong solution has been taken out of the ore, a weak stock solution is run on the top of the ore to a depth of about 6½ in. The cock connecting with the vacuum cylinder is then opened, and in about thirty hours the second solution has passed through; after which about 10 in. of water is run onto the top, and when this has gone through the ore the operation is completed. The sludge-door in the vat is opened, and the sand sluiced out by means of two 2 in. hose-pipes under a head of 150 ft. The vats are all circular, 22 ft. 6 in. in diameter, and 4 ft. in depth, of which 5 in. is taken up by the filter bottom, which consists of a wooden grating with edges rounded off on the upper side, having a strong Hessian cloth laid over the top, which acts as a filter. The vats are made of kauri timber, 3 in. in thickness; the bottom is held together by six

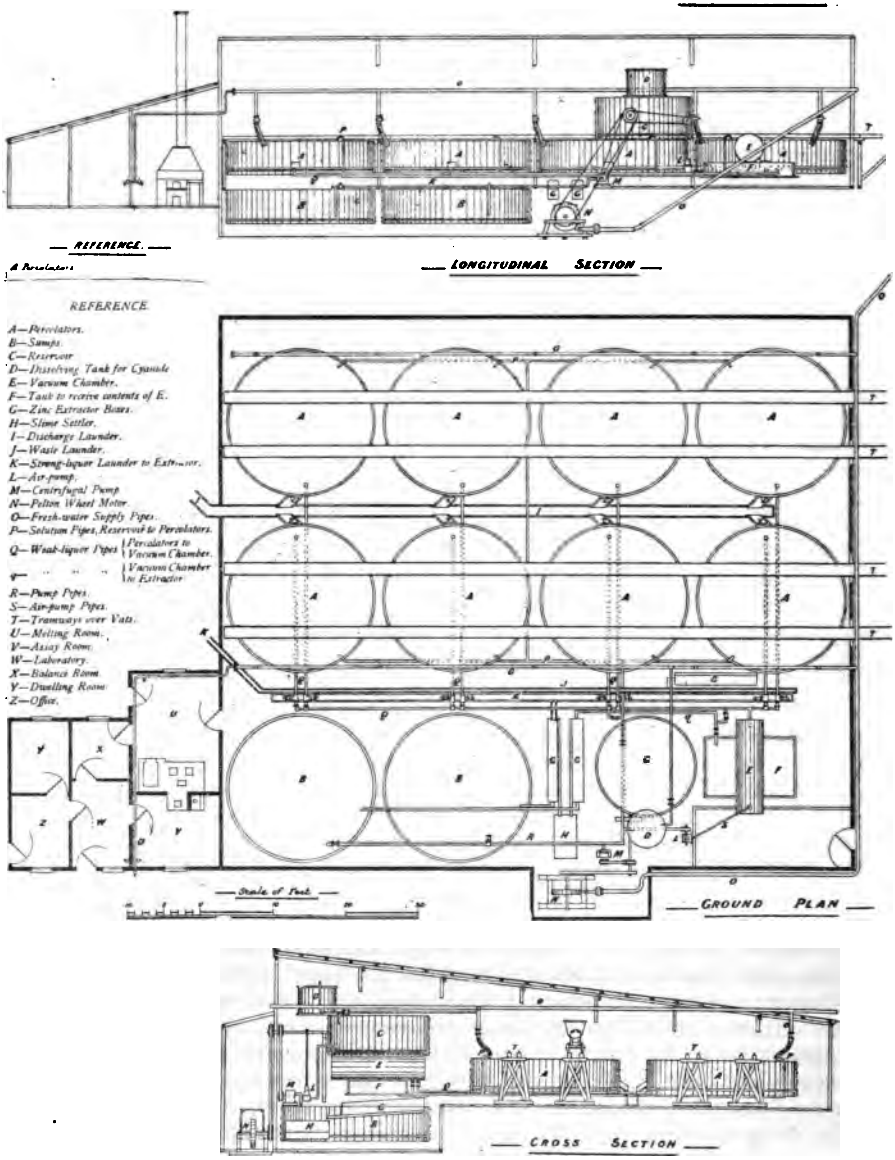
bolts of $\frac{3}{4}$ in. diameter. The staves are about 3 in. in width, joined close, having the bottom rebated into the sides. Each vat is held together by five round-iron hoops, three of which are $\frac{7}{8}$ in. and two 1 in. in diameter, having three turn-buckles on each hoop. The plant consists of thirteen of these vats and two sumps of the same diameter, but 6 ft. in depth. Each vat holds 30 tons of ore for treatment; and it takes about four days to fill a vat, treat the ore, and have it ready for filling again. The precipitation boxes are 16 ft. long, 2 ft. deep, and 17 in. wide, divided into twelve divisions, of which the first and last are sand filters, to clean the solution going in, and to prevent any gold slimes from being washed out.

"The cost of treatment is being reduced every month; at the present time it amounts to about $3 25 per ton. This includes drying, milling, treatment by cyanide, and all expenses, except the royalty paid to the owners of the MacArthur-Forrest patents, from the time it leaves the mine-hopper until the bullion is in bars."

The tailings from the former pan-amalgamation process are being worked in a cyanide plant erected for that purpose by the Cassel Gold Extraction Company. These works were completed about the end of February, 1894. Mr. H. A. Gordon gives the following description, illustrated by plans which I reproduce: "The works are situated in a hollow below the tailings dam, so as to allow the tailings to be run at a good grade into the percolation vats, and from there to be discharged by sluicing, without the necessity for any lifting or rehandling.

"The building has a frontage of $116\frac{1}{2}$ ft., and is 77 ft. in breadth, and includes laboratories and offices, situated in a 'lean-to' at one end and communicating with the main building. The plant consists of eight circular percolation vats 20 ft. in diameter and 4 ft. in depth (internal measurement), arranged in two rows, and having an intermediate discharge-launder, toward which the vats have a slope of 2 in. to facilitate the flow of solutions and the sluicing-out of residues. All the vats are built of specially selected and well seasoned heart-of-kauri, the timbers being 3 in. thick. The sides are hooped with $1\frac{7}{8}$ in. iron bolts, connected and screwed up by nuts and cast-iron boxes, there being three boxes to each ring. The bottom planks are bolted and dowelled tightly together independently of the sides. The filters at the bottom consist of a foundation of 2 by 3 in. slats, 9 in. apart, covered by 1 in. molding, which supports the canvas strainer. This filter is very easily laid, and is most effective in practice. Each vat is provided with a cast-iron door 18 by 12 in., fixed at the bottom of the side near the discharge-launder, for the sluicing of residues. There are two sumps of same size and design as the percolators, and situated between the percolators and front of the building, and on a sufficiently low elevation. The sumps are floored over. In the same line are placed the reservoir and cylindrical vacuum chamber, 13 ft. by 3 ft. 9 in., under which latter is provided a small rectangular tank, 12 ft. by 8 ft. by 18 in. deep, capable of holding contents of vacuum chamber. The reservoir is 13 ft. 9 in. diameter by 5 ft. deep (inside measurement), and is at such an elevation as to permit solutions to flow therefrom into percolators. There are three extractor boxes, 12 ft. 8 in. by 19 in., with side discharge for slimes and a settler for cleaning-up. The tank for dissolving the cyanide is an iron pan about 3 ft. 6 in. in diameter by 2 ft. 6 in. in height, and is capable of dissolving four boxes—*i. e.*, 1,000 pounds—of cyanide per

TAILINGS CYANIDE WORKS ERECTED BY THE CASSEL GOLD EXTRACTING C° L^{TD}
AT WAIHI N Z.



day. It is so arranged that the requisite amount of strong solution may be run into the reservoir by simply turning a handle. A 4 in. centrifugal pump serves for returning the solution from the sump to the reservoir, and also an 8 in. vacuum pump, which is capable of producing a vacuum of 26 in. of mercury. A line of pipe runs along above each row of percolation vats, with a connection at each tank for the hose and nozzle. One man can empty a vat containing upward of 40 tons in two hours. A tramway connects the tailings-dams with the works, and two sets of lines run over the top of each vat, so that the tailings may be equally distributed without the necessity for handling. The chief characteristic of the plant is its extreme simplicity and the easy access to any portion of it; the absence of any subdivisions or partitions within the main building exposes the whole of the plant constantly to the eye of the operator.

"The system employed of running the solutions into parallel launders instead of pipes, enables the solutions from each vat to be separately and readily sampled and any mishap may be at once detected. The usual method of procedure is as follows: Side-tipping trucks are run from the tailings-pit over the top of the vat which is to be charged. The contents of the trucks are tipped onto cross-bearers resting on struts, which serve to break the fall of the tailings, and to divide them equally over the bottom of the vat. Both tramway and bearers are supported entirely independent of the vats, so that no vibrations may be communicated to the latter. A charge consists of sixty-five truck-loads—about 33 tons, dry weight—and as soon as the vat is full, 'strong' solution—about 6 tons of 0.7 per cent—is run onto the top from the elevated reservoir. Provision is made for either upward or downward percolation, but the latter is usually adopted. The solution is now permitted to gravitate through the mass of the charge, and to eventually percolate through the false bottoms into the series of launders in which it is conducted to No. 1, No. 2, or No. 3 zinc precipitation box, according to its strength in bullion and cyanide. About twenty-four hours after the 'strong' solution, about an equal amount of 'weak' solution (0.25 per cent) from the sumps is pumped on and allowed to gravitate. The residues are now washed with about 10 tons of water in two charges, which are rapidly drawn off by suction, and which displace the 'weak' solution and leave the residues free of either dissolved bullion or cyanide. The solutions run from the zinc boxes to the sumps, whence they are pumped to the reservoir or percolation vats, to be used over again for sluicing or weak solutions as required. A clean-up of the gold in the zinc boxes takes place fortnightly."

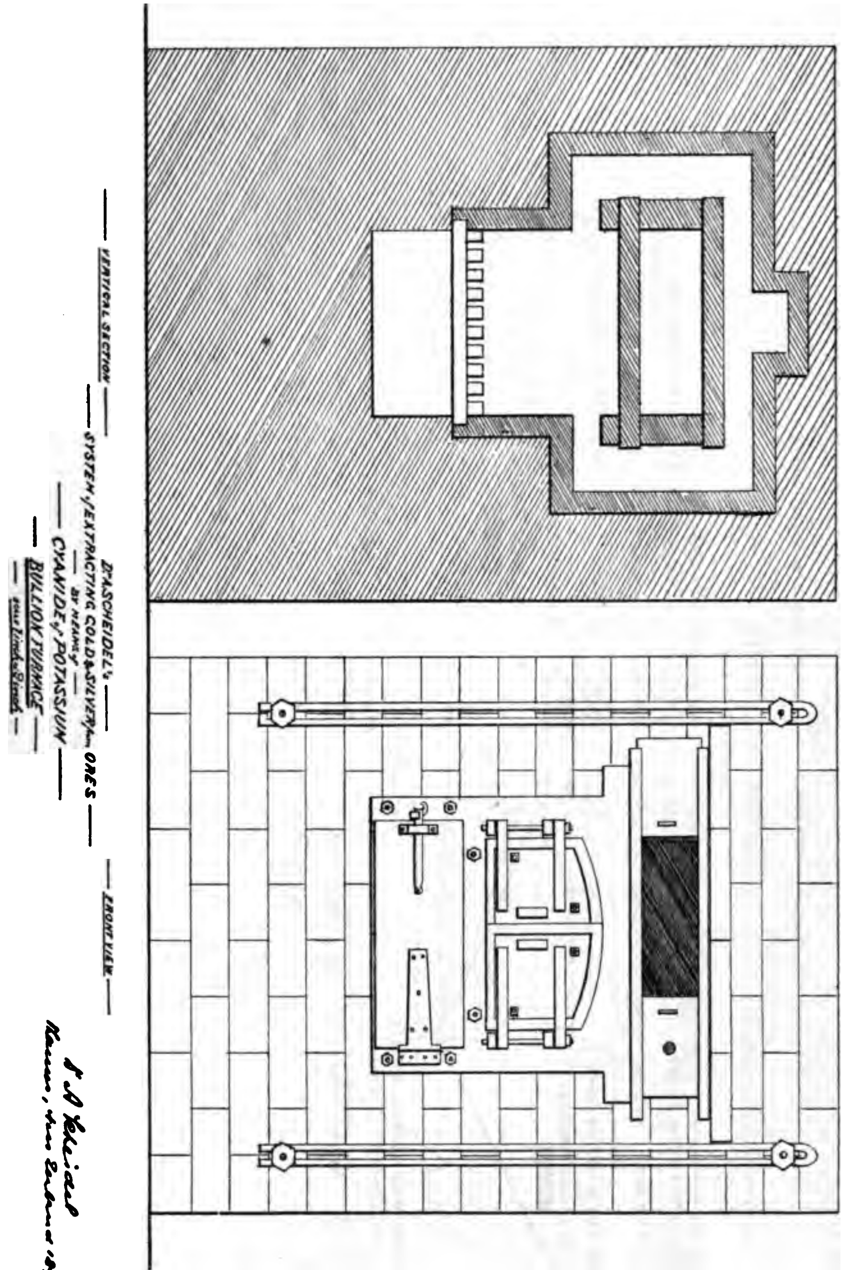
The New Zealand Mining Report of 1893 contains the description of an interesting experiment which was made some time ago in the Waihi works, with an apparatus by which ore was intended to be rapidly extracted by a cyanide solution acting in a jigging motion on the ore in iron cylinders. The cylinders used were, however, too long and narrow, containing as they did some 10 ft. in depth of ore, which the solution had to be forced through. The effect of this was that the solution could not be made to percolate through the whole of the ore, but passed up between the cylinder and the ore, the solution being forced into the cylinder by a pump, at a pressure of 100 lbs. per square inch. This pressure should have been sufficient to force the solution through, but as the pulverized material offered greater resistance than the contact

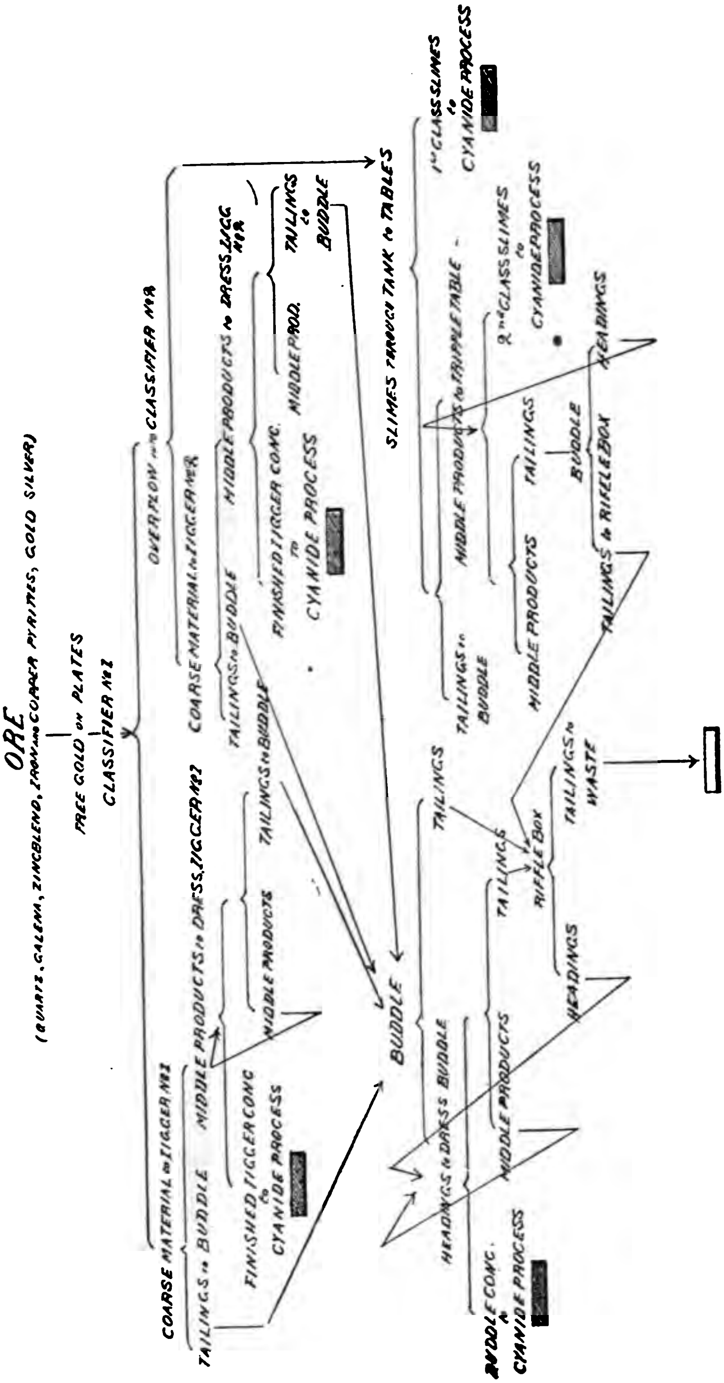
between the material and the side of the cylinder, the solution went through the weakest spot, and had little effect on the ore. The process ("the Bohm Process") proved a failure.

The unquestionable success of the percolation process with the Crown and Waihi mines has led to its adoption by a number of other companies which treat either ores or tailings by the process, as the Te Komata and Waiorongomai mines at the Upper Thames, and two or three companies on the Kuaotunu gold field. The Kuaotunu ore, in which the gold is exceedingly fine, is especially adapted for the treatment, the only difficulty experienced—a mechanical one—is caused by the amount of slimes formed by some of the ores, which interfere with filtration. The plants on that field do not offer any special point of interest; they are of small extent and give satisfactory results. The best one, that of the Tryfluke Company, will, however, be described on account of the attempts made therewith to run the tailings direct from the battery into the percolation vats. The plant consists of four working tanks, each 12 ft. wide, 16 ft. long, and 4 ft. deep, having a filter-bed of 3 in. in thickness, covered with a coarse cloth. The depth of ore in the tanks is about 3 ft. 6 in. and about 6 tons of 0.25 per cent cyanide solution are used per charge. This is what is termed the strong solution. The tap, which allows the filtrate to flow away, is so regulated as to take about 24 hours for that purpose. After flowing through six compartments of a filter box, which are filled with fine zinc-turnings, the solution passes into a sump, 18 ft. long by 14 ft. wide and 4 ft. deep, from where it is pumped into the tank again, thus forming the second solution. This latter is allowed to filter through as fast as possible, and, after going through the boxes filled with zinc, it flows into another sump of the same dimensions as that already mentioned. The ore is then washed with pure water, after which the material is shoveled out of the tanks and run onto the waste dump. The second solution in the sump, previously referred to, is pumped into a reservoir placed at a higher level than the working tanks. This reservoir is 10 ft. long, 8 ft. wide, and 5 ft. deep. The solution is made up to the required strength before again being used. The company tried to run the tailings directly into the tanks from the battery, but they, like others, found that the amount of slimes in the ore prevented the cyanide solution from filtering, and they are now making arrangements to run the tailings into a large pit, from which they will be lifted into the cyanide tanks. (Extract from N. Z. Gov. Mg. Rep.)

All works so far referred to are situated in the North Island; on the large gold fields of the South Island the process has not found more than experimental application. Experiments have been made with gold-bearing cement from the extensive deposits on the west coast, where almost inexhaustible quantities of conglomerates, containing black magnetic oxide of iron and very small quantities of gold, are found. Such experiments were made, for instance, in the Reefton School of Mines, by treating the cement in lumps, but they were not always successful, apparently for mechanical reasons. When the cement is crushed, a very good percentage is said to be extracted, the gold being fine and well suited for the purpose. Tests with tailings from the Inangahua River gold fields have given good results, and a plant for working a considerable deposit of tailings is nearing completion at Boatman's.

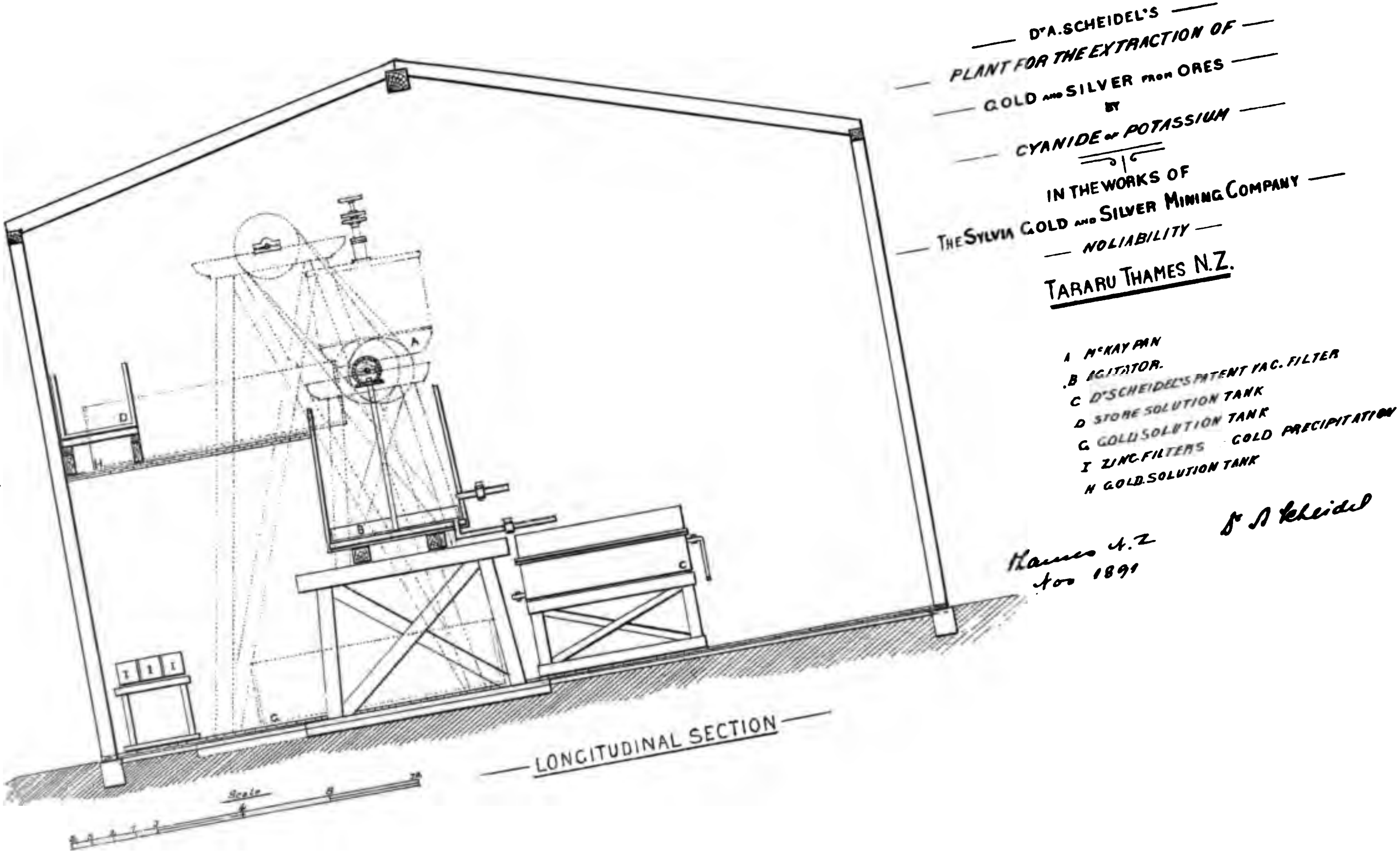
The only instance in New Zealand where the agitation system has

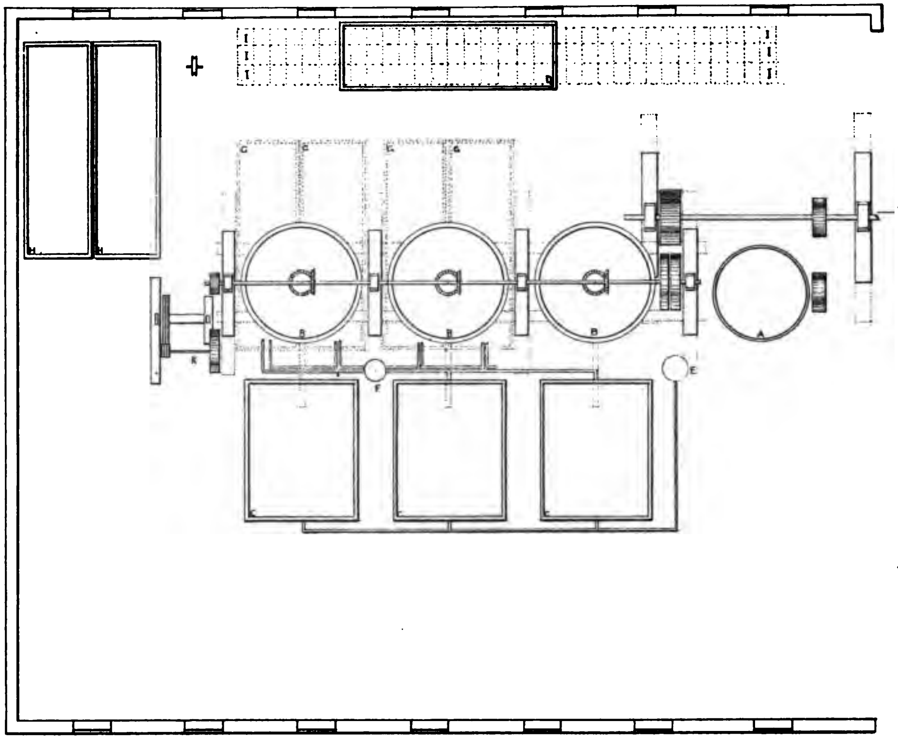

VERTICAL SECTION
DR. A. SCHEIDEL'S
SYSTEM of EXTRACTING GOLD & SILVER from ORES
BY MEANS of
CYANIDE of POTASSIUM
BULLION FURNACE
FRONT VIEW




S. A. Scheidel

THE CONCENTRATING AND CYANIDE PROCESS IN THE WORKS OF THE SYLVIA GOLD AND SILVER MINING COMPANY. TARARU, NEW ZEALAND.


Dr A. SCHEIDEL'S
PLANT FOR THE EXTRACTION OF
GOLD AND SILVER FROM ORES
BY
CYANIDE OF POTASSIUM
IN THE WORKS OF
THE SYLVIA GOLD AND SILVER MINING COMPANY
NOLIABILITY
TARARU THAMES N.Z.
A McKAY PAN
B AGITATOR.
C Dr SCHEIDEL'S PATENT VAC. FILTER
D STORE SOLUTION TANK
G GOLD SOLUTION TANK
I ZINC FILTERS GOLD PRECIPITATION
H GOLD SOLUTION TANK
Thames N.Z
Nov 1891
Dr A Scheidel
LONGITUDINAL SECTION
Scale




— Dr A. SCHEIDEL'S —

— PLANT FOR THE EXTRACTION OF —

— GOLD AND SILVER FROM ORES —

BY

— CYANIDE OF POTASSIUM —

IN THE WORKS OF

THE SYLVIA GOLD AND SILVER MINING COMPANY —

— NO LIABILITY —

TARARU THAMES N. Z.

A McKAY PAN
B AGITATORS
C Dr SCHEIDEL'S PATENT VAC. FILTER
D STORE SOLUTION TANK UPPER FLOOR
E
F GOLD SOLUTION PUMP
G GOLD SOLUTION TANKS GROUND FLOOR
H GOLD SOLUTION TANKS UPPER FLOOR
I ZINC FILTERS FOR GOLD PRECIPITATION
K FRICTION GEAR OR LIFT

Thames N.Z.
Nov. 1891.

Dr A. Scheidel

— Scale —

— PLAN —

been worked on a large scale is in the works of the Sylvia Company in Tararu, Thames, where concentrates of a very complex character have been treated with full success by the author. The ore of the mine named contains, in the deeper levels, a high percentage of galena, zinc-blende, and copper and iron pyrites. Most of the bullion is contained in the sulphurets and cannot be saved by amalgamation, nor is there any opportunity for smelting the sulphurets after concentration or for remunerative exportation of the same. After successful trials with the concentrates subjected to cyanide treatment, I constructed an agitation plant, of which I append my plans, reproduced from the N. Z. Gov. Mg. Rep. 1892. The concentrates in question are classed as jigger concentrates, first-class slimes, second-class slimes, and buddle concentrates. They are named in the order as they are obtained during the dressing process. The finest products contain the most galena; they are the richest in gold and silver, and give the highest percentage of extraction. A detailed description of plant and extraction process is given in the mining report above referred to. I give here only the most salient points: The plant, constructed of wood (kauri pine), consists of three agitators, 6 ft. in diameter by 6 ft. in depth; three Scheidel's vacuum filters (patented) of 35 ft. square filter surface each; the necessary solution tanks, pipes, and pumps, and bullion roasting and melting furnaces. The experience gained with this plant, which works well with the exception of the faults which necessarily adhere to the use of wood for cyanide plants, induced me to construct, later on, the plant for the Utica Mine, California, of steel. This plant, being free from the only fault, that above mentioned, answers the purpose to perfection. The results of extraction of the Sylvia concentrates vary in accordance with the quality of the material, the slimes giving generally better results than the coarser products, and the richer first-class slimes return a higher percentage of gold and silver than the second-class slimes, which are of lower grade. Eminently satisfactory results have been obtained from the best slimes, from which as high as 96.45 per cent of the gold and 94.59 per cent of the silver assay-value have been extracted. The average extraction of 100 tons amounted to 86.11 per cent of the gold and 67 per cent of the silver; corresponding with 85.22 per cent of the total value. The extraction of the least suited material, the jigger concentrates, coarse and low grade, amounted to 80.32 per cent of the gold and 50 per cent of the silver. The average extraction of all classes of concentrates amounted to 82.67 per cent of the assay-value. The total amount of bullion extracted by me from about 300 tons of material amounted to upward of $51,000. The time of agitation and the strength of solution vary in accordance with the quality of the material. The quantity of cyanide used for the highest grade ore amounted to less than 1 per cent, and for low-grade material considerably below 0.5 per cent of the ore. The time of agitation varied between six and twenty-four hours. The method of working the plant, which permits the treatment of twenty tons per twenty-four hours, is identical with that of the Utica plant, which I propose to describe in detail. The Sylvia Company enjoyed, on account of the plant being the pioneer plant of its kind, special privileges in reference to royalty for the use of the reagent.

New Zealand is among the few countries, outside of Africa, where cyanide treatment of ores, tailings, and complex concentrates has been in all instances a perfect success. Many of the ores of the Colony are

particularly suited to the treatment. Its still more extensive application is prevented by the royalty charged by the owners of the MacArthur-Forrest patents, in reference to which fact I quote the Government Inspecting Engineer of the Colony, Mr. H. A. Gordon, who says (Gov. Min. Rep. 1893):

"There is no gainsaying the fact that the cyanide of potassium is a good reagent for gold, and undoubtedly the best agent for extracting gold, especially where the latter is in a very finely divided state among the ore; but the royalty charged is prohibitive, and it will never be largely used until it is lowered. If the Cassel Company (the owners of the MacArthur-Forrest patents) would have been content with about 2 per cent royalty, the process would have been almost universally adopted by every company in New Zealand."

The following particulars represent the results of the working of the cyanide process on the New Zealand gold fields up to the end of December, 1893:

| Names of Companies. | Bullion, in oz. | Bullion, value. |
|---|---|---|
| New Zealand Crown Mines, Karangahake, from ore | 15,064 | $145,930 |
| Sylvia Gold Mg. Co., Thames, from concentrates | 14,552 | 51,965 |
| Tryfluke Gold Mg. Co., Kuaotunu, from tailings | 3,072 | 35,895 |
| Great Mercur Gold Mg. Co., Kuaotunu, from tailings | 605 | 6,410 |
| Te Aroha Gold Mg. Syndicate, Waiorongomai, from ore and tailings | 1,981 | 4,595 |
| Te Komata Gold Mg. Co., Upper Thames, from tailings | 1,741 | 5,155 |
| Red Mercury Gold Mg. Co., Kuaotunu, from tailings | 149 | 1,230 |
| Silverton Gold Mg. Co., Waihi, from tailings | 299 | 2,105 |
| Waihi Gold Mg. Co., Waihi, from ore | 1,097 | 9,045 |
| Welcome Gold Mg. Tailings, Boatman's, from tailings | 200 | 2,255 |
| Totals | 38,760 | $264,585 |

The returns by cyanide during the twelve months ending March 31st last are: Bullion obtained from ore, 14,774 oz.; from tailings, 12,478 oz. The year 1894 promises to be even a more successful one, six new cyanide plants being in course of erection. The bullion obtained by cyanide during the quarter ending 30th June amounted to 13,030 oz. from ore, and to 7,073 oz. from tailings. Fifty-two per cent of the total bullion product in the North Island of New Zealand was produced during that period by the cyanide process.

In the other colonies of the Australasian group the process is comparatively slow in being introduced. The monetary crisis that has prevailed in the colonies, combined with prejudice and skepticism on the part of the mining community, have retarded its introduction.

(b) **Tasmania.**—In this colony the process has not yet been introduced on a working scale. So far, the only use that has been made of it, has been on a small and experimental scale, and in a very imperfect manner. (Letter from Government Secretary for Mines, January 15, 1894.)

(c) **Western Australia.**—Here the process has not been adopted on a large scale, "the mining industry being yet in its infancy. The process is, however, considered the best known for Western Australian gold ores, and some minor experiments have been made with tailings, giving good results." (Letter from Government Secretary of Mines, anuary 19, 1894.)

**(d) South Australia.**—This colony has several plants for the working of the cyanide process, one of which, that at Mt. Torrens, is being worked as a custom works by the Mines Department of the State for the purpose of giving the mine owners an opportunity to have their ores tested. A Government plant undoubtedly inspires miners and prospectors with confidence. A charge just sufficient to cover cost of treatment is made by the department.

"The plant has now been in operation for some weeks, and the treatment of an ore parcel from the Blacksnake Mine has been completed. The ore contained on an average 16 dwts. 15 grs. of gold; of this, 10 dwts. 8 grs. were saved by the battery, 4 dwts. 9 grs. by cyanide, leaving 1 dwt. and 22 grs. in the tailings. The ore contained quartz, hematite, and about 4 per cent of iron pyrites." (Letter from Secretary for Crown Lands, May 23, 1894.)

A plant for treating 500 tons per month is at work on tailings at the Virginia Gold Mining Company's property, and is doing good work. The tailings from the battery are allowed to dry, and are then trucked into vats of a capacity of between 25 to 30 tons, 16 ft. in diameter by 5 ft. in depth. The sumps are built of cement, 16 ft. by 14 ft. by 12 ft. The consumption of cyanide is about 1¼ lbs. per ton of tailings. The tailings before treatment assay from 10 to 15 dwts., after treatment, 1 dwt. 7 grs. per ton. The bullion is refined with nitre. The vats are charged twice a week. (Government letter.)

**(e) Queensland.**—At Charters Towers, a plant capable of treating 800 tons per month has been erected by the Australian Gold Recovery Company, Lim., the owners of the MacArthur-Forrest patents for Australia, from which the following information has been obtained: It is a custom plant. The chief material treated is sludges, which are purchased from the surrounding mills in varying quantities. These sludges are concentrates which have been submitted to fine grinding and amalgamating in berdans; this material is, if necessary, mixed with coarse sand or tailings, and treated by percolation; vacuum pumps are used to assist. Different classes of ore of a refractory character are treated in the works, and the conditions of treatment are varied with the character of the ore. Operations were started in August, 1892, since which date about 9,200 tons of sludges have been treated, with a return of 9,633 oz. of gold. "At Croydon a tailings plant has been erected at the Cumberland property of a capacity of 1,500 tons per month. Plants for the treatment of 2,000 and 1,000 tons per month are in course of erection for the Croydon Quartz Crushing Company and the Pioneer Gold Mining Company, respectively." (Australian Gold Recovery Company.)

**(f) New South Wales.**—In this colony the cyanide process is at work only at the Mitchell's Creek gold mine, where the plant and operations are described by Mr. James Taylor, the Government metallurgist, as follows: "The works have been erected for the treatment of a dump of old tailings estimated to contain about 18,000 tons, and found by careful sampling to contain 8 dwts. 4 grs. of gold and 11 dwts. 10 grs. of silver per ton. The plant consists of two 400-gallon iron tanks, two storage vats, six percolating vats, two sets of ten precipitating boxes, two sumps, iron pipes for conveying the solu-

tions and washes, side-tipping wagons and tram-rails for charging and discharging the percolating vats, steam boiler and pump to return the cyanide solution from the sump to the storage vat, small muffle furnace for roasting the precipitated gold with its admixture of zinc and copper, laboratory and assay furnaces.

"In the two iron tanks the cyanide stock solution is prepared by dissolving crude potassium cyanide, suspended in a wire gauze tray in water. The crude salt contains about 75 per cent of pure potassium cyanide, and the stock solution is made up to a strength of from 10 to 25 per cent. From the iron tanks the solution is run through a canvas-bottomed box, which serves as a filter, into one or the other of two storage vats, as it is needed to bring up the strength of the returned liquor to the standard required for the treatment of the next charge of ore, say about 0.7 per cent of cyanide. These two wooden storage vats are 16 ft. in diameter and 5 ft. deep, and are placed sufficiently high to discharge into the next series of vats by means of iron pipes. The six large percolating vats, each 18 ft. in diameter and 5 ft. deep, are provided with filter-bottoms, built up by laying ribs of wood, notched on the under side, along the bottom of the vat at regular intervals. On these ribs is spread cocoanut matting, and over this comes a layer of wool-pack. The edges of the filtering cloths are well caulked along the sides of the vat. The vats are charged with tailings by means of side-tipping wagons, carried on an over-head tram-line, and the exhausted tailings are discharged by being shoveled through an 18 in. hole in the side of the vat, near the bottom, into a wagon running on rails and leading to the waste dump. Each percolation vat receives a charge of 35 tons of tailings, and two vats are emptied and refilled daily during the day shift, so that 70 tons of tailings are treated every twenty-four hours. The fresh charge of tailings is first soaked with returned solution, which begins to make its appearance at the bottom of the vat in about three hours from the time of its application. A solution of 0.7 per cent is then turned on, and this is allowed to act for about twenty-five to thirty hours; it is then drawn off, running direct to the precipitating boxes. About twenty inches of liquid from the storage vat is sufficient to soak the charge, and a similar amount of the reinforced solution is enough for the gold extraction, after the application of which water is run on to wash out the cyanide solution. Each of the percolation vats can be supplied with cyanide solution from the storage vats by means of a 2½ in. pipe, and with water through a 2 in. pipe, for washing after the cyanide solution has been drained off, or before the cyanide solution has been added should that be found necessary, as is sometimes the case. The tailings are under treatment in the vat during a period of about sixty-four hours from the completion of charging to the commencement of discharging the vats.

"An inch and a half pipe proceeding from the bottom of the percolating vats takes the gold solution to the precipitating boxes, where the liquid is caused to pass upward successively through a series of ten boxes, filled with fine zinc turnings, made on the spot from the zinc linings of the boxes in which the potassium cyanide has been imported. The gold and silver are precipitated upon the zinc as a fine black slime, accompanied by any copper that may have been taken up by the solution from the tailings. After passing these boxes, the solution, almost or altogether free from gold, is collected in a couple of sumps, each 16 ft. in

diameter and 5 ft. deep, from which it is pumped back to the storage vats, ready for another application, either for soaking, or, when suitably reinforced, for gold extraction. When a clean-up is being made, the zinc in the boxes is well stirred to shake off the slimes, which are then washed out through a plug-hole into a launder, where they are collected, and either roasted and melted with a little flux, or treated with acid and then melted; the mode of procedure depends upon the amount of base metal mined with the gold and silver. All solutions going from the percolating vats pass through the precipitating boxes, excepting of course the wash water retained as moisture in the tailings when the vat is emptied. The tailings contain, as noted, 8 dwts. of gold and 11 dwts. 10 grs. of silver to the ton. The returns for a recent run of ten weeks show that 64 per cent of the gold and about the same of silver was actually recovered; but by assay of the tailings after treatment it appears that 70 per cent of the gold had been removed; hence something like 6 per cent seems to be locked up in the plant, and may be obtained later. The cost of cyanide during the same period of ten weeks amounted to $2 58 per ton of tailings, and the total cost of treatment to $3 38 per ton. The tailings contain from 0.25 per cent to 0.50 per cent of copper, the presence of which increases the amount of cyanide used by from 75 cts. to $1 per ton, and further acts injuriously, as it is precipitated by the zinc with the gold and silver and debases the bullion obtained, thus necessitating refining operations. The force employed in the works consists of one scientific manager, two men on alternate twelve-hour shifts to attend to the circulation of the solutions, nine men to fill and empty the vats, one man to prepare the cyanide solution and to do odd jobs, and a laboratory boy. The cost of the plant was approximately $10,000; it was erected under the auspices of the Australian Gold Recovery Company, Limited" (MacArthur-Forrest patents). During six months 9,972 tons have been treated, with an extraction of 2,512 oz. of gold.

**(g) Victoria.**—"A plant of 2,000 tons per month is being constructed at the New Golden Mountain Gold Mining Company's property, and it is proposed to treat the ore at these works directly by cyanide, without any previous treatment or battery amalgamation". (The Australian Gold Recovery Company.)

## C. The United States of America.

The introduction of the cyanide process in the United States as a metallurgical process on a commercial basis has so far been slow; the territory is vast, the mining districts are widely scattered, and there is an almost invincible prejudice against any new process, particularly patent processes, owing to the innumerable failures in which for years past much money has been lost. Of late the process has been tested on many ores from almost all mining camps in the United States. It has been found successful, on a small scale, in many instances. Its technical application has, however, not always been the expected success, for which fact various causes are responsible. In many instances plants have been erected where either the ore is unsuited for the process or where the supply of suitable ore is insufficient; in other instances the working of the process has been intrusted to incompetent hands, which

naturally led to a failure. The MacArthur-Forrest patents and the Simpson patent are owned in the United States by the Gold and Silver Extraction Company of America, Lim., of Denver. A number of extraction works have been erected all over the United States by mining companies and other parties. To obtain reliable information of their results and their plants has proved possible only in a few instances, and it is hardly possible to form from such information an adequate idea about the success of the process in this country.

(a) **Utah.**—The first mill to operate cyanide treatment in this Territory is owned by the Mercur Gold Mining and Milling Company. That company had just completed a pan-amalgamation plant at a cost of \$30,000, which proved a failure, only 20 per cent of the gold being recovered, when small tests with cyanide, followed by good results, led to the introduction of the process on a large scale, which has since proved a full success. For the following description, by Louis Janin, Jr., I am indebted to the Engineering and Mining Journal (Oct. 7, 1893):

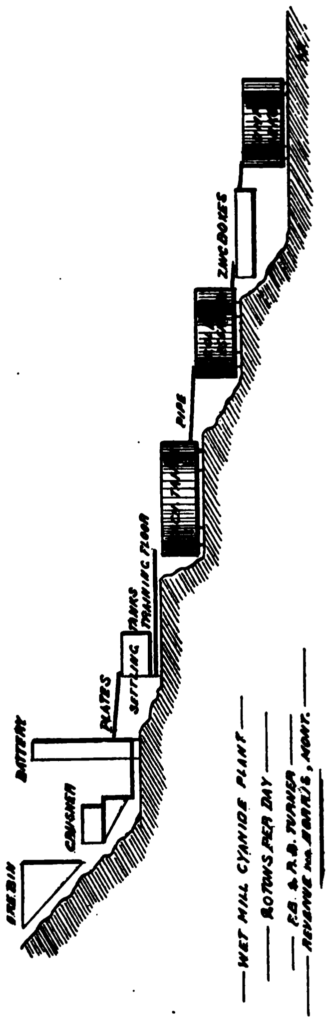
"The ore passes through a Dodge rock-breaker, and is crushed by two sets of Wall's corrugated rolls; the first are set to one half inch, the second to one quarter inch. This very coarse ore (over 20 per cent of the product which goes to the leaching vats does not pass a half-inch mesh) is treated by cyanide percolation. The dimensions of the vats are 12 ft. 8 in. diameter; depth to false bottom, 35 in.; giving a capacity of about fourteen tons when the vats are filled to within 6 in. of the top. In consequence of the crudeness of the crushing, the time of leaching varies greatly; it occupies between ten and two hundred and forty hours. It is claimed that the ore (which is a silicious surface-ore with the gold finely divided) is singularly constant in value and quality; the wide differences in the time of treatment are ascribed to the differences in mechanical condition. As a rule, the cyanide solution is left standing with the ore for twelve hours; it is then passed through continuously, until practically all gold is extracted; the time required varies from thirty-six to forty-eight hours. The percolation liquor passes through zinc boxes 40 ft. long, and is returned to the stock solution tank, where its proper strength is made up again by addition of cyanide. After the ore in the tank has been leached sufficiently, the tank is allowed to drain. However, a considerable quantity of solution, about 400 lbs. to the ton, or, with the 0.25 per cent solution used, about 1 lb. potassium cyanide to the ton of ore, remains in the vat. To dislodge this, wash water is used, either plain water or sometimes a weak solution resultant from washing. In the latter case, the weak solution is stored in separate tanks, and this arrangement allows washing with a minimum wastage of solution. The extraction of the Mercur ore has varied; at the beginning of operations it was considerably below 70 per cent, but as experience with the process increased the results became better; the average is now given as between 85 and 90 per cent. The cost of treatment during an early period of the work is given at \$2 40 per ton, divided as follows: Potassium cyanide (1.27 lbs. per ton), 66 cts.; zinc (0.55 lb. per ton), 5 cts.; labor (seven shifts per twenty-four hours, 6 day and 1 night), \$1 12; supplies, repairs, fuel, and freight, 57 cts.; total (not including office expenses, royalty, and superintendence), \$2 40. Since that period, the expenses have been reduced as the amount of cyanide lost per ton of ore has been diminished, and a larger quan-

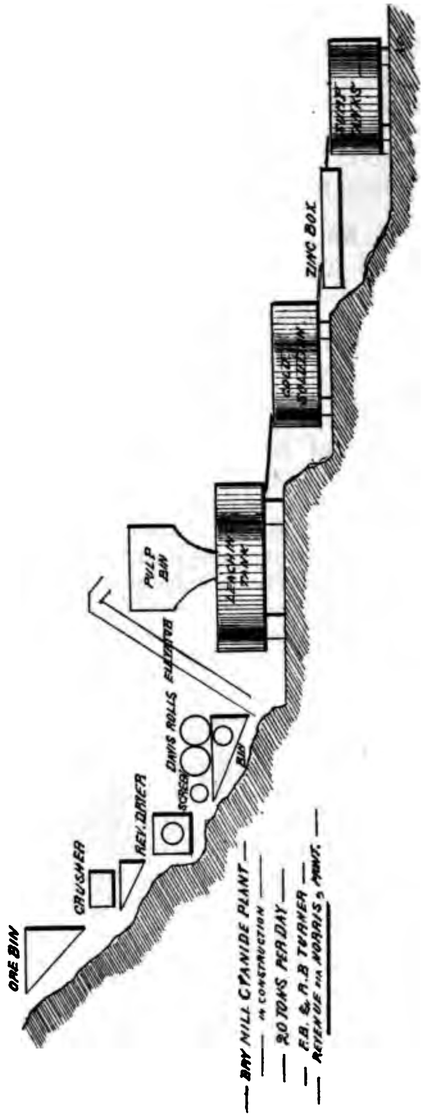
tity of ore is reduced with the same amount of labor. Comparative results by actual experience on the ore by amalgamation and cyanide treatment are as follows: 1,500 tons of ore treated by amalgamation gave an average extraction of 20 per cent at a cost of $4 25 per ton for milling; 1,600 tons of ore, treated during 90 days by cyanide, averaged an extraction of 88.5 per cent, at a cost of $2 25 for milling." The introduction of the cyanide process has made the Mercur Company a remarkable financial success; it paid during the first five months of this year $150,000 in dividends. The plant is being increased to a capacity of 250 tons per day.

**(b) Montana.**—One of the first cyanide mills in this State was erected by F. B. & R. B. Turner, in Revenue, Madison County, who supplied the information which follows: The gold in the Revenue ores has always been very hard to save, the best amalgamation methods only saving from 25 to 27 per cent of the assay-value; the application of the cyanide process increases the returns to from 80 to 87 per cent. Extensive tests have proved that the wet-crushing and cyanide treatment is profitable only on low-grade ore, as the loss of low-grade slimes connected with that method is considered immaterial. The most successful treatment with cyanide has been found to be percolation of dry-crushed ore. The present plant (see diagram, p. 86) is used for wet-crushing of low-grade ore. The tailings from the battery pass into settling pits, and the slimes are allowed to flow into a large reservoir below the mill for eventual future treatment. The tailings settled in the pits are shoveled into the leaching tanks; the running of the pulp direct into the percolation vats proved here, like in many other localities, a failure, owing to the slimes. A dry-crushing plant of 25 tons per day is now in course of construction (see diagram); after its completion, ores of about $25 value will be crushed in the dry, poorer ores in the wet mill. The Revenue ore is nearly pure silica, containing from 1 to 2 per cent of iron peroxide and no sulphurets. It is crushed and passed through a 30-mesh steel screen. Some of the percolation vats are of 10 ft. diameter by a depth of 4½ ft., others are of 12 ft. diameter by 4 ft. deep; they are made of 3-in. Oregon pine. The strength of the solution varies from 0.6 to 1 per cent; percolation takes from 24 to 36 hours, after which the solution, which is circulated or pumped back, varies from 0.4 to 0.8 per cent of cyanide. One half ton of solution, on an average, is used for treating one ton of ore. The total extraction amounts to from 80 to 87 per cent of the assay-value of the ore, 27 per cent of which is obtained from amalgamation on the plates. Cyanide extracts from 73 to 79 per cent of the value left in the tailings after amalgamation. No difficulty is being experienced in precipitating the gold. The consumption of zinc amounts to about half a pound per ton of ore. Sulphuric acid is used for bullion refining; the fineness of melted bullion is from 920 to 940. The total cost of treatment amounts to $5 per ton, including $1 patent-royalty, and consists of the following items: Crushing and labor, $2; chemicals, $2; patent-royalty, $1. The actual consumption of cyanide is from 2½ to 3 lbs. per ton of ore. The total cost of the plant amounted to about $20,000, including engines, boilers, stamps, and vats, all placed in position. Eight to ten men are employed in the mill.

A cyanide mill erected by the Henderson Mountain Mining and

Milling Company, near Cooke, Park County, was worked on surface hematite ore for some time with fairly satisfactory results.




(c) **Colorado.**—The owners of the MacArthur-Forrest patents possess a small plant for testing ore parcels in South Denver.

The Cripple Creek Gold Extraction and Power Company erected a plant in Cripple Creek, in reference to which I received the following communication from the technical manager (Mr. J. K. Turner):

"A small plant has been a remarkable success, and an addition to it is now in course of erection, bringing its capacity up to 50 tons daily. The machinery will finally consist of three Gates' crushers, two pulver-

izers, four screens, four iron leaching vats of 20 ft. diameter, two solution tanks of 15 ft. diameter, and four zinc boxes of 40 ft. length. The plant is running as a custom mill, and many classes of ore are being treated; at present about 30 tons per day. The ore is crushed dry and passed through a 20-mesh screen. The cyanide solution used is 0.75 per cent, the strength of which is by percolation reduced to 0.5 per cent. One half ton of solution is required for the treatment of one ton of ore. The extraction of gold averages 90 per cent, that of silver 84 per cent. Sulphuric acid is used for bullion refining; the bullion is from 775 to 790 fine. Copper and zinc compounds in some ores have been found to interfere with extraction. The average value of the ore is $30 before and $3 after cyanide treatment. The cost of treatment amounts to $4 70 per ton. The cost of the plant was $20,000. Seven men are employed in the works."

The successful cyanide treatment of Cripple Creek ores, as here described, is very interesting, for the ores in that district contain a large amount of telluride minerals. Many ores contain sylvanite, krennerite, and calaverite.

The Puzzler Gold Mining and Milling Company of Denver have ceased to work their cyanide plant at Ward, Boulder County, although they were successful with the process.

A cyanide plant at Junction Creek is reported in successful operation. (M. S. P.)

(d) **Nevada.**—A branch company was formed by the owners of the MacArthur-Forrest patents for introducing the cyanide process on the Comstock Lode. A number of laboratory experiments made in the Con. Virginia and California Company did not lead to the adoption of the process.

(e) **Arizona.**—No information can be obtained in reference to the process in this Territory, where a company has been organized for its introduction, with the exception of that obtained from the Champies Mine, Yavapai County, where the results were unsatisfactory, apparently on account of faulty technical manipulation.

(f) **New Mexico.**—The Deep Down Mine had adopted the cyanide process, but abandoned it in favor of pan-amalgamation, the reasons for which are explained (Engineering and Mining Journal) as consisting in a change of the character of the ore.

(g) **South Dakota.**—The Black Hills Gold and Silver Extraction Mining and Milling Company of Deadwood have of late erected a cyanide plant, about which the general manager, Mr. I. S. Childs, gives the following details:

"The ore consists of from 80 to 95 per cent of silica, accompanied by variable quantities of iron, both in the form of peroxide and the various conditions of partial oxidation, and contains traces of copper, manganese, arsenic, and antimony. The ore is dry-crushed with rolls, and passes through a 30-mesh screen. The percolation vats of steel are 24 ft. in diameter by 3 ft. in depth. The cyanide solution is usually of 0.5 per cent strength; its quantity amounts to half the weight of the ore. From 85 to 90 per cent of the assay-value in gold and from 50 to 75 per cent

of the silver value is extracted. All extracted bullion is recovered. Lime is used with the ore and caustic soda with the concentrates to remedy the 'acidity.' The consumption of zinc amounts to 0.55 lb. per ounce of bullion recovered. The treatment takes from twenty-four to forty-eight hours. The value of the material before treatment is $20; the tailings assay from $2 to $3 in gold. The total expenses for treatment, including $1 patent-royalty, are $3 50 per ton. The plant for the treatment of 40 tons per twenty-four hours cost $25,000. Fifteen men are employed per twenty-four hours."

The Golden Reward Chlorination Works, of Deadwood, are reported as adding cyanide works to their plant. The works are custom works.

Many tests with cyanide have been made in various other States, in some instances leading to the adoption of the process on a commercial scale; in others, experiments were conducted with little knowledge and in a cursory manner; in other instances, again, the character and qualities of the ore prevented cyanide treatment from being a success. One instance, where the process was first used but afterwards abandoned, is the Creighton Mining and Milling Company, Cherokee County, Georgia, where the extraction from concentrates from old oxidized tailings, as described by the general manager, was reasonably good, being 82 per cent of the assay-value; from the fresh concentrates from deeper levels, however, the returns were only 50 per cent, in consequence of which the process was discarded and barrel chlorination introduced in its place (H. T. Fisher). In other instances the value of the ore in the mine fell off, so that not only the cyanide process, but all operations had to be stopped. An instance in point offers—the Moratock Mine, Montgomery County, North Carolina. The engineer in charge stated (Engineering and Mining Journal) that he had no trouble in treating the ores of that mine by cyanide and making a high extraction (95 per cent) at a moderate cost. Cyanide solution of 0.25 per cent was used; the total cost of mining, milling, and royalty amounted to about $3 75. The mine, however, was shut down, the ore-value receding to $1 25.

(h) **California.**—The Shasta Gold Recovery Company erected a mill under the direction of Mr. A. B. Paul. The mill has been working for some time successfully. Mr. Paul was one of the first, if not the first, who used wet-crushing of the ore with cyanide solution, instead of water, in the mortars. No authentic information could be obtained in reference to his results.

A cyanide plant has been erected at the Gold Run Mine, Siskiyou County, the working of which has been described in the California Mining Report, No. 11, page 430.

The following information referring to the cyanide process in Kern County, has been contributed by Mr. W. L. Watts, assistant in the field to the California State Mineralogist: "A company which was organized in St. Louis to reopen the Bright Star Mine in Kern County, attempted to work several thousand tons of tailings at that mine by the cyanide process. It is said that the leaching plant used for the purpose could only handle six tons of tailings every twenty-four hours, and that about 45 per cent of the precious metal was recovered."

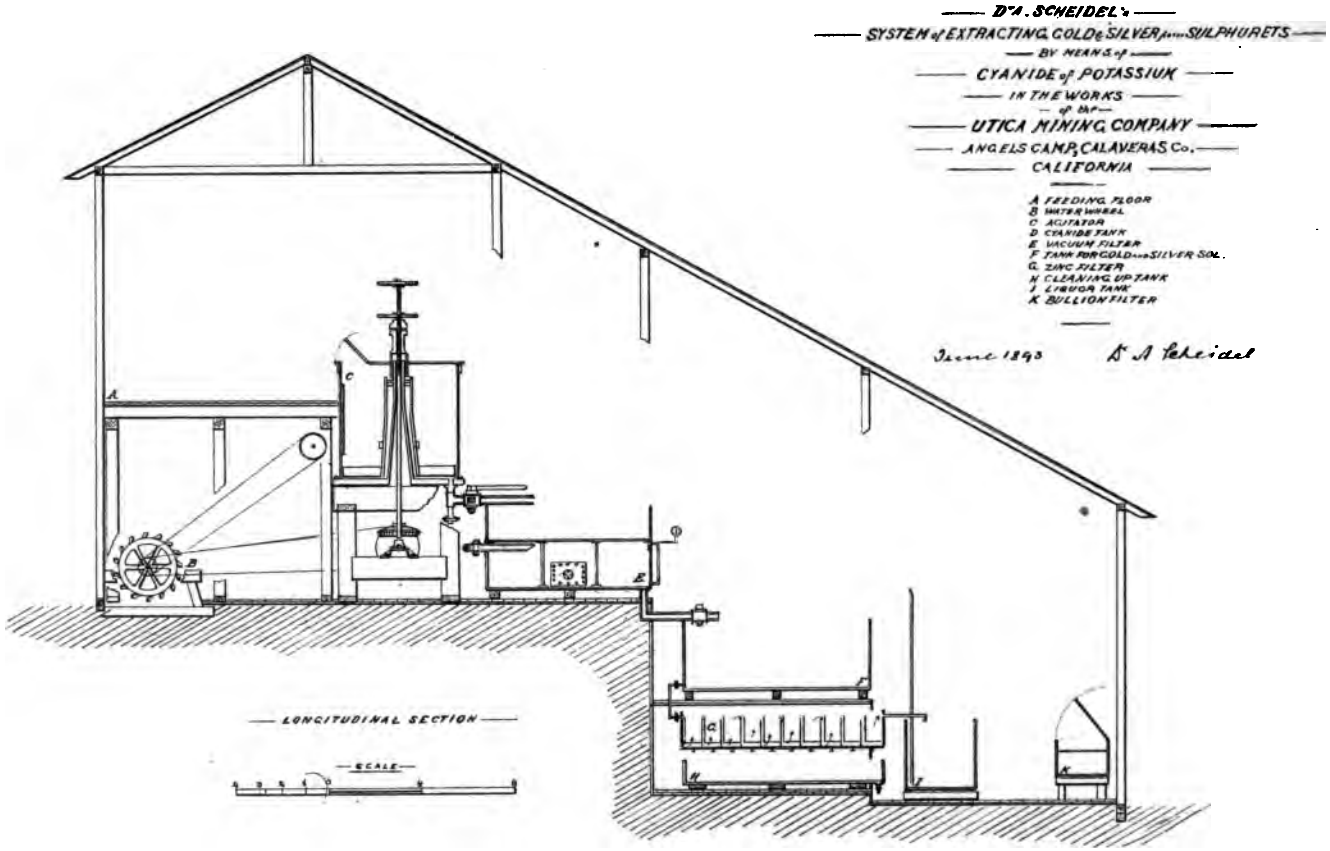
"*The Cyanide Process at Havilah, Kern County.*—In 1892, Messrs. Stebbins and Porter commenced leaching at Havilah. They first treated

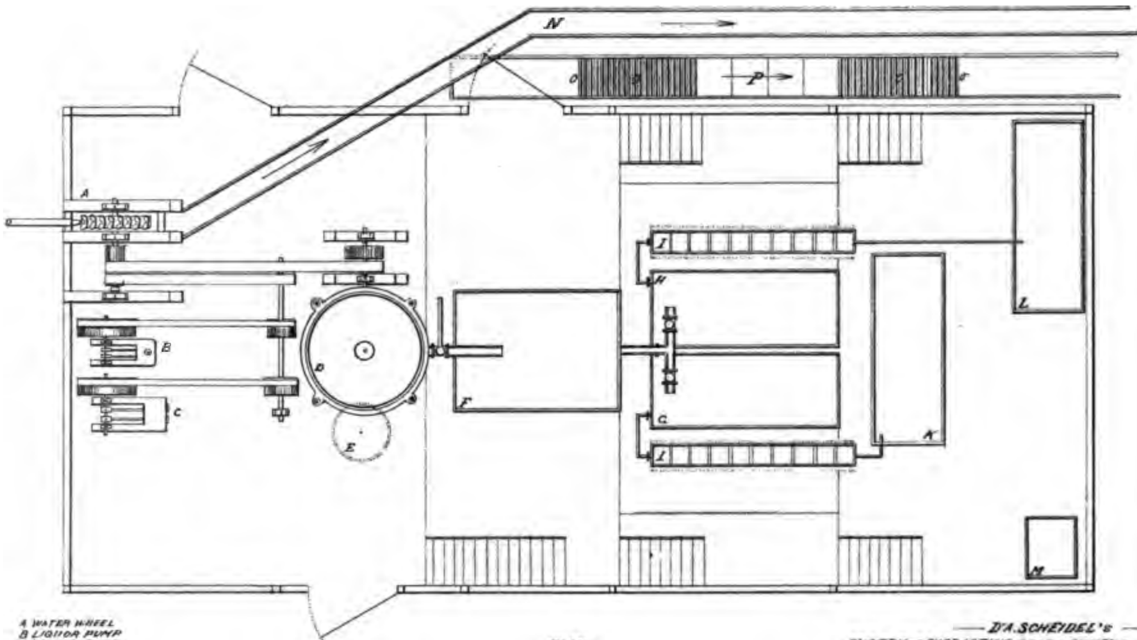
50 tons of heavy sulphuretted concentrates at the Reese mill. The assay-value of these sulphurets ranged from $15 to $48 per ton. Owing to leakage and other causes, their first experiments were not remunerative, but the last and richest portion of this batch of concentrates was treated with great success, 86 per cent of the precious metal being saved. One difficulty encountered arose from the fact that, owing to the sulphurets having long been exposed to the air, they were partially oxidized into acid sulphates; the detrimental effect of these sulphates was eventually overcome by neutralizing them with quicklime. One hundred tons of tailings were then treated at the Hayes mill. These tailings showed an assay-value of $20 per ton, and about 91 per cent of the precious metal was saved. This last lot of tailings consumed 3 lbs. of potassium cyanide to the ton of ore. The process employed is as follows: The tailings are first treated in a tank 6 ft. long, 6 ft. wide, and 6 ft. deep, and holding a charge of six tons of ore. In this tank the tailings are saturated with a solution containing one half of one per cent, by weight, of potassium cyanide. The tailings are allowed to stand in this solution for forty-eight hours, and are then sluiced into 'filter-bottom tanks,' where they are washed. These 'filter-bottom tanks' are 7 ft. long, 10 ft. wide, and 3 ft. deep. The tailings are sluiced from the saturating tank with a stock solution which has a strength of one-fifteenth of one per cent of potassium cyanide. About 1,000 gallons of stock solution are used in sluicing six tons of ore. The filtrate is then tested to determine whether a sufficient percentage of the gold and silver value of the pulp (as shown by assay) has been dissolved. If the filtrate is found to contain a sufficient percentage of the gold and silver, the solution is drawn off and the tailings are washed with fresh water; enough wash-water is used to replace the amount of the solution taken up by the tailings. The amount of water thus added to the solution is usually about 500 gallons to six tons of ore. If on testing the solution it is not found to have dissolved a sufficient percentage of the gold and silver, the stock solution is left on the ore until all the gold and silver which it is possible to extract by this process has passed into solution. The temperature of the solution employed has varied from 40° to 80° Fahr., and within these limits no difference was experienced in the solubility of the gold and silver. In this process it was found to be far better to assay the filtered solution than the macerated pulp, for a perfect volumetric sample of the filtrate can be readily obtained. Test assays—four assay-tons of the auriferous solution of cyanide of potassium are evaporated, and the resultant auriferous compound is mixed with half an assay-ton of litharge, and fluxed with glass. The assay is conducted as an ordinary assay for gold and silver."

"In speaking of the different ores they have treated by the cyanide process, Messrs. Stebbins and Porter state that it has been their experience that the majority of ores showing free gold also contain gold in any sulphide which may be present; but that, except in the case of sulph-arsenides, ores showing no free gold seldom contain auriferous sulphides. In September, 1893, Messrs. Stebbins and Porter were building a mill in which to treat ore by the cyanide process at the Iconoclast Mine."

A cyanide agitation plant was erected by me (the author) in 1893, at the Utica Mine, Calaveras County, which, being successful in all details, shall here be described in full. The Utica Mining Company are the owners of an extensive milling plant, consisting of 160 stamps with Frue vanners and Tulloch concentrators, and a canvas plant for the saving

of the sulphuret slimes which escape from the concentrators. Up to last year all concentrates were extracted by the Plattner chlorination process, which treats the coarse vanner-pyrites well; the fine slimes of the canvas plant, however, do not give equally satisfactory results, on account of their containing a very large percentage of carbonate of lime, which is troublesome and involves loss in the roasting furnace on account of its fineness and lightness, and is costly in chlorination on account of its taking up chlorine. The slimes are also difficult to leach in the chlorination vats. I examined the different classes of concentrates as to their fitness for treatment with cyanide. The coarser sulphurets from the concentrators did not give, without further grinding, sufficiently high extraction results to warrant the substitution of the chlorination by the cyanide process. The results of the experiments with the slimes from the canvas plant, which were very satisfactory, led to the construction of the present agitation plant. This plant (see illustration) is completely built of steel and iron, and consists of the following parts: A vertical cylindrical agitator (constructed by Mr. C. D. Lane and myself), 5 ft. in diameter by 5 ft. high, of $\frac{1}{4}$ in. steel plate, with a cast-iron bottom 2 in. thick, with strengthening ribs underneath; to the bottom is cast a cone, through which passes the vertical shaft, which carries four arms. To these are attached the four paddles of $\frac{1}{8}$ in. steel, 6 in. wide, twisted like the plates of a propeller. A ring connecting the four paddle arms gives greater stability to them. The shaft with the paddles can be raised, by means of a screw-spindle, 4 ft. above the bottom of the apparatus. A wrought-iron ring, 3 in. wide and $\frac{1}{2}$ in. thick, riveted outside around the top of the agitator, strengthens the structure. The driving gear is placed below. An opening, 4 in. diameter, in the bottom, discharges the contents of the agitator through a pipe, furnished with a stopcock, onto Scheidel's patent vacuum filter, placed on the floor below and in front of it. Here a perfect separation of the cyanide gold solution is effected from the residues. This filter (see diagram) is built of $\frac{1}{4}$ in. steel, with bottom $\frac{3}{8}$ in. thick; it forms a rectangular box 3 ft. 6 in. deep, 7 ft. long by 5 ft. wide; 2 ft. above the bottom is a perforated steel filter-bottom of $\frac{3}{8}$ in. boiler plate, made in three movable sections, supported by angle-iron running around the sides, and by the vertical support of double T iron. The perforations are of $\frac{1}{2}$ in. diameter, at a distance of $\frac{1}{2}$ in. from each other. This filter-bottom fits closely to the sides of the apparatus; it is covered with a blanket, which is kept in position by bars running along the four sides, and fastened by thumb screws. A grating in three sections of $\frac{3}{8}$ in. round iron serves to protect the cloth; the intervals of 3 in. between the bars are filled in with coarse sand. The filter partition divides the apparatus into two compartments, one above the other; the lower forms a closed box, which is in connection with a duplex vacuum pump, by means of which the air can be rarefied when the filter-bottom is covered with pulp. The upper part, above the filter-bottom, receives the contents of the agitator. The real bottom of the apparatus has a discharge with a 3 in. stopcock, for running off the filtered solution into either one or the other of the two solution tanks, which are standing on the floor one step lower, in front of the filter. All cocks and taps of the plant are of considerable diameter, which secures a quick charge and discharge. The filter is provided with a gauge to indicate the height of the solution within, a gauge indicating the degree of vacuum, an air-tap to permit influx of air when the filtered solution is being discharged, and a manhole.


Dr. A. SCHEIDEL'S
SYSTEM of EXTRACTING GOLD & SILVER from SULPHURETS
BY MEANS of
CYANIDE of POTASSIUM
IN THE WORKS
of the
UTICA MINING COMPANY
ANGELS CAMP, CALAVERAS Co.
CALIFORNIA
A FEEDING FLOOR
B WATER WHEEL
C AGITATOR
D CYANIDE TANK
E VACUUM FILTER
F TANK FOR GOLD and SILVER SOL.
G ZINC FILTER
H CLEANING UP TANK
J LIQUOR TANK
K BULLION FILTER
June 1893
Dr A Scheidel
LONGITUDINAL SECTION
SCALE




A WATER WHEEL
B LIQUOR PUMP
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F VACUUM FILTER
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H TANK for 2nd GOLD SOLUTION
I ZINC FILTERS
K TANK for 1st LIQUOR
L TANK for 2nd LIQUOR
M BULLION FILTER
N TAIL RACE
O RIFFLES
P SILVER PLATES

— PLAN —

— SCALE —

— DR A. SCHEIDEL'S —
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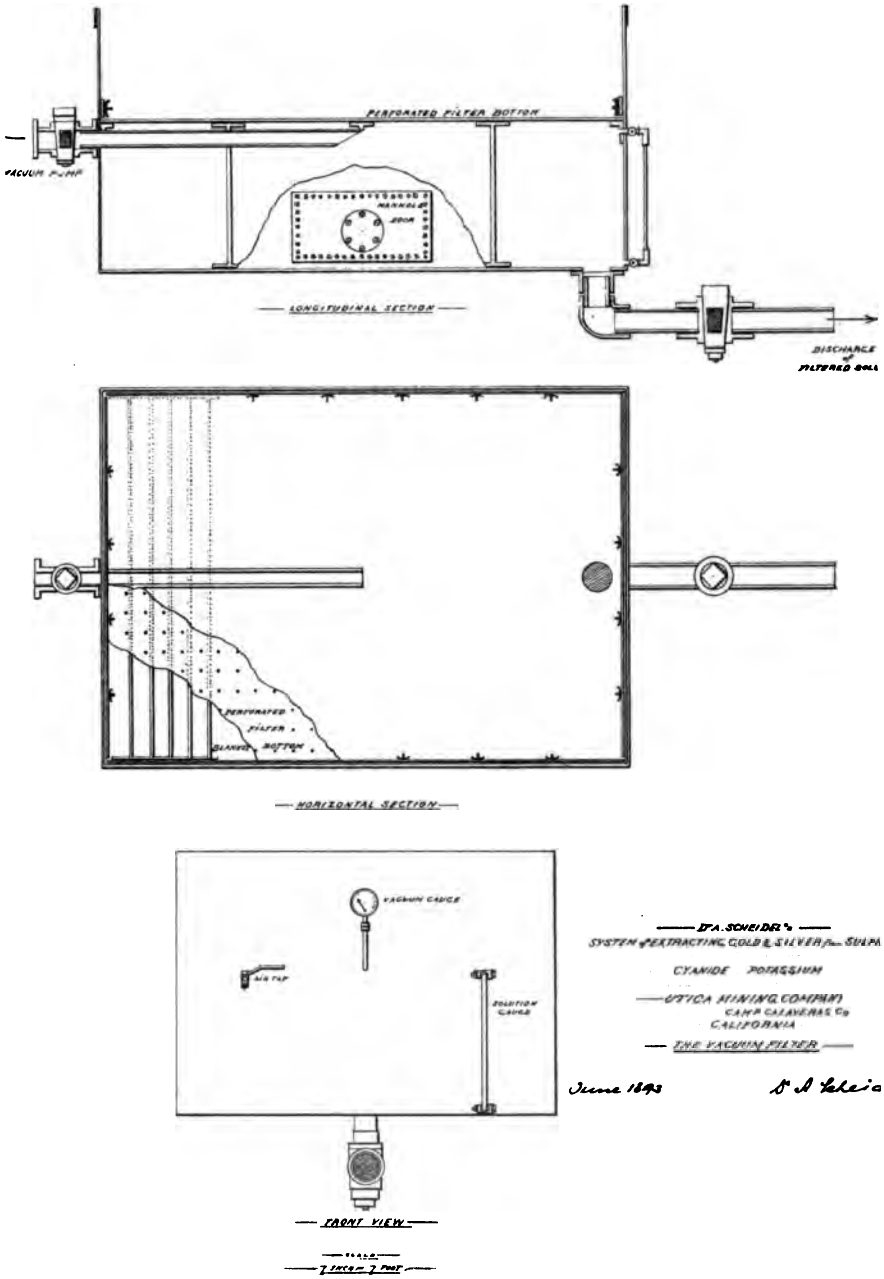
June 1893 — A. Scheidel


PERFORATED FILTER BOTTOM
VACUUM PUMP
MANHOLE DOOR
LONGITUDINAL SECTION
DISCHARGE of FILTERED SOL.
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HORIZONTAL SECTION
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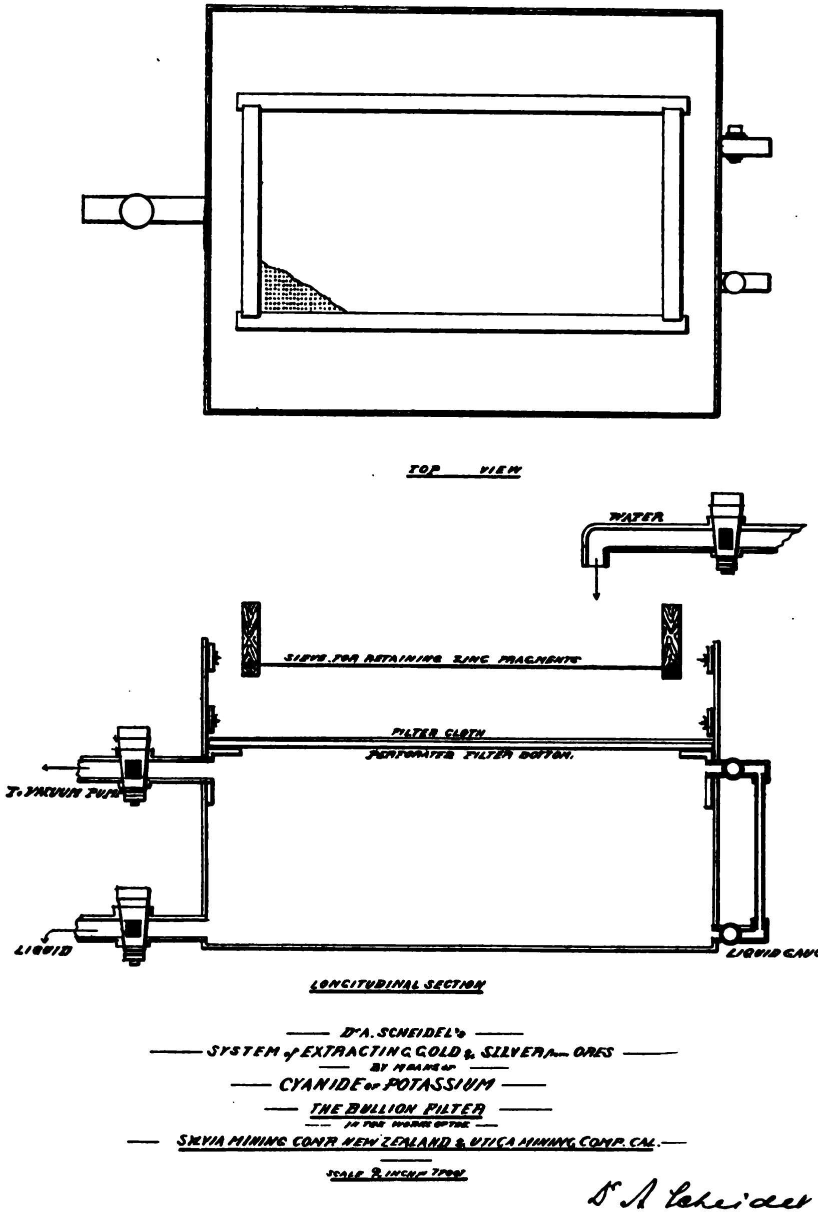
THE VACUUM FILTER
June 1893

The mode of working the plant is this: The cyanide solution is charged into the agitator, the paddles are set in motion by revolving the shaft, the ore is charged by degrees, and the agitation is kept up for the required time, after which the pulp is discharged from the agitator onto the filter. The vacuum pump is then set in motion, and filtration under the influence of atmospheric pressure will at once commence. The solution will soon be sucked through; then washing follows, first with liquor from former operations, which has already passed through the zinc boxes, and finally with clear water. These operations of filtering and washing take about two hours. It is advisable to suck the tailings as dry as possible before each new wash is put on, which permits the complete removal of the gold solution with a very small amount of liquid, one half ton of which is sufficient for washing a charge of two tons of ore. If the filtration has been properly managed, no degree of continued washing can improve the results. The filtered solutions are clear. The first or original solution, together with the first wash, will be run off into one of the two solution tanks in front below the vacuum filter; the following washes run into the other. These tanks are 8 ft. long by 3 ft. wide and 3 ft. deep, made of $\frac{1}{4}$ in. steel. Each of the tanks is in connection with a zinc precipitation box, 9 ft. long by 21 in. deep and 9 in. wide, divided into ten compartments; there is an interval of 1 in. between each two compartments. The false perforated steel bottoms of the chambers, which can be removed if desired, are $2\frac{1}{2}$ in. above the true bottom of the box (see diagram, p. 31). The bottom of the box has a number of 1 in. iron faucets, one corresponding with the center of each filter compartment; the sides of the box are 4 in. higher than the partitions within, which insures absolute safety against the liquid running over the sides of the box, if one or the other compartments should become blocked. The gold solution flows into the box through a 1 in. cock, enters the first compartment from below through the perforated false bottom, percolates the zinc shavings placed thereupon, leaves it, and enters the second, and so forth. A steel settling tank, 12 in. deep, 12 in. wide, and 9 ft. 3 in. long, is placed below the precipitating box for receiving the bullion when cleaning up (see general demonstration of process, above). The zinc used consists in turnings of $\frac{1}{800}$ in. thick, turned from cast zinc cylinders on a lathe; 2 lbs. fill one compartment. The solution passes through the box at the rate of 700 gallons in twenty-four hours. The bullion precipitation of the solution is very efficient as it passes from compartment to compartment, which amounts to passing ten times through a zinc column of 14 in. high by 9 in. square. As shown by the following table of analysis,

| One Ton of Liquid contains— | Gold. | | | Silver. | | |
|---|---|---|---|---|---|---|
| | Oz. | Dwts. | Grs. | Oz. | Dwts. | Grs. |
| Originally | 5 | 14 | 0 | 2 | 4 | 8 |
| After 14 in. of zinc column | | 16 | 1 | | 5 | 3 |
| After 28 in. of zinc column | | 5 | 9 | | 1 | 10 |
| After 42 in. of zinc column | | 2 | 14 | | | 17 |
| After 56 in. of zinc column | | 1 | 4 | | | 12 |
| After 70 in. of zinc column | | 1 | 1 | | | 11 |
| After 84 in. of zinc column | | | 22 | | | 3 |
| After 98 in. of zinc column | | | 15.43 | | | 0 |
| After 112 in. of zinc column | | | 14.96 | | | 0 |
| After 126 in. of zinc column | | | 13.36 | | | 0 |
| After 140 in. of zinc column | | | 12.34 | | | 0 |




The solution leaving the zinc box contains only 12.34 grs. of gold, or 0.0045 per cent of its original contents, and only 3 grs. of silver, or 0.0028 per cent of the original silver value. Simultaneously the solutions were analyzed for available cyanide, but no decrease in the strength of the solution, which remained constantly at 0.3185 per cent, could be ascertained. "At another period I studied the solubility of zinc in

cyanide solution, of which I give the following figures: 0.2634 grs. of filiform zinc were submerged in 50 cc. cyanide solution of 0.26 per cent; after seven days of frequent agitation these were reduced to 0.2584 grs., and after fifty-six days to 0.2252 grs., which means that after seven days 1.98 per cent, and after fifty-six days 14.47 per cent of the zinc were dissolved. From this observation it follows that the loss of cyanide in the precipitating boxes, by means of its being taken up by zinc, is sometimes overestimated." The washes pass through a similar precipitating box. The liquids, when leaving these boxes, go as liquor No. 1 and liquor No. 2, into tanks of the same size as the solution tanks, from where a pump will deliver them wherever wanted. Liquor No. 1 serves for making up the new solution for the next charge; liquor No. 2 is used for washing purposes on the vacuum filter. No liquor ever leaves the works; the quantity in circulation remains stationary. The bullion obtained from the zinc boxes is passed, as formerly described, through a sieve onto the bullion vacuum, which itself is a miniature reproduction of the vacuum filter (see diagram). It has the following dimensions: Length, 2½ ft.; width, 2 ft.; total depth, 1 ft. 6 in. The perforated filter-bottom is fixed 12 in. above the true bottom. The bullion is very slimy; in fact, it is the more slimy the freer it is from zinc; its filtration and washing take some time. When the mass is tolerably dry, it is put into a wooden tub and treated with diluted sulphuric acid. The heat of the reaction I have always found sufficient to make the operation a speedy and satisfactory one; the bullion is then permitted to settle, the liquid is siphoned off through the bullion filter, and the solid matter is washed by decantation with water. This washing process is continued until all soluble salts are removed. The bullion is then partially dried on the filter, and finally dried in a small muffle furnace; complete drying of the bullion by artificial heat before the acid treatment is not advisable. The thoroughly dry bullion is pulverized and well mixed with soda and borax, and melted in a plumbago crucible as described before. The bullion thus obtained is 946 fine; the slag is clean, it contains the usual few granules of bullion, but, freed from them, does not give any assay results. The bullion could be still further refined, but to no commercial advantage. The steel of the tanks has not as yet shown any effects from cyanide, nor does it exercise any influence on the solutions. All apparatus is composed of plates and sheets riveted together; leakages, if any, can be easily stopped by a varnish made of asphaltum dissolved in bi-sulphide of carbon. As mentioned, this plant has been constructed for the treatment of slime concentrates from the canvas plant; such concentrates contain a varying percentage of carbonate of lime, in some instances as much as 95 per cent, which, however, does not interfere mechanically or otherwise with their satisfactory extraction by cyanide. Such conditions would make chlorination all but impossible, as alluded to before. For agitation the material requires an amount of solution equal to 30 per cent of its weight, and six hours of time. The described plant is capable of treating a much larger amount of slimes than are usually produced per day by the canvas plant; its services are therefore only periodically required. The average consumption of cyanide, calculated from a large tonnage of slimes treated, amounted to 4.3 lbs. per ton, costing $2 27; the labor amounts to $1; the total expenses of treatment by cyanide to $3 50 per ton. The average extraction amounts to 93.18 per cent of the gold, and

90 per cent of the silver assay-value; as high as 96.57 per cent of the gold has been extracted in some instances. The extraction of the gold during the agitation goes on as shown by this table:

| Treatment of Slimes by Agitation. | Gold per Ton. | Extraction—Per Cent. |
|---|---|---|
| Sample before treatment | $88 00 | |
| Sample after 1 hour's agitation | 13 00 | 85.23 |
| Sample after 2 hours' agitation | 11 00 | 87.50 |
| Sample after 3 hours' agitation | 7 00 | 92.05 |
| Sample after 4 hours' agitation | 7 00 | 92.05 |
| Sample after 5 hours' agitation | 6 00 | 93.18 |
| Sample after 6 hours' agitation | 5 00 | 94.31 |
| Sample after 7 hours' agitation | 5 00 | 94.31 |
| Sample after 8 hours' agitation | 5 00 | 94.31 |

Within the first hour 85.23 per cent of the gold are extracted; during the following five hours the increase of extraction is slow and irregular; after six hours no further extraction takes place. For experimental purposes I continued agitation up to twelve hours without improving on the result. The treatment of the slime concentrates by agitation was preferred on account of its quicker, cheaper, and better results, as compared with percolation. All Utica concentrates, as in fact all concentrates I ever had to deal with, contain a small amount of amalgam, part of which is found on the bottom of the agitator; part of it leaves the works with the tailings, and is recovered in Hungarian riffles and on amalgamated silver plates; part of the mercury goes undoubtedly into the cyanide solution, and is precipitated with the bullion in the zinc boxes.

Other sulphurets, such as the Frue vanner concentrates of the Utica, Madison, and Eureka mines, were treated on a more or less extensive scale by the same plant; results were, however, not very satisfactory on account of their coarseness. All sulphurets of the Utica Mine are pure sulphide of iron. The fine canvas-plant concentrates, although less clean, are as a rule richer in gold; their extraction averaged 93.18 per cent, whereas the vanner concentrates gave only 81.38 per cent, which, although reasonably good at the rate of $4 per ton cost of treatment, cannot compete with a chlorination treatment, which yields 90 per cent of a $50 ore at a cost of $6 50. (The large size of the Utica chlorination works offers special advantages and permits chlorination at this figure, which is much lower than the cost anywhere else in California.) A large number of tests proved that a high percentage of the gold is contained in the coarser particles of the sulphurets; this will account to some extent for the comparatively low percentage of cyanide extraction. It would lead too far to give here the results of the large number of experiments in reference. The experiments were extended to roasted concentrates after their reduction to uniform size, which gave on a small scale excellent results (98.09 per cent extraction). I am indebted to Colonel Hayward, Mr. Charles D. Lane, and Mr. James Cross, of the Utica Mine, for their permission to publish the diagrams and the described results of the cyanide works which I erected for them. The cost of the plant is divided as follows:

| | |
|---|---|
| Grading and foundations | $200 00 |
| Building | 300 00 |
| Shafting, belting, and putting into place | 135 00 |
| Agitator | 260 00 |
| Vacuum filter | 165 00 |

| | |
|---|---|
| Three tanks | $160 00 |
| Two zinc boxes | 260 00 |
| Two steel tanks | 85 00 |
| One vacuum pump | 235 00 |
| One liquid pump | 130 00 |
| Pipes, stopcocks, faucets, etc. | 70 00 |
| Total | $2,000 00 |

The Standard Consolidated Mining Company, Bodie, California, started a 100-ton cyanide plant (50-ton vats) on September 17th for the treatment of tailings.

The amount of ores, fine concentrates, and tailings suitable for cyanide treatment is considerable in this State. There is no doubt that a great amount of gold is now being lost with the slimes in most mills here, as elsewhere, and in many instances dry-crushing of the ore and direct cyanide treatment would vastly increase the returns. Coarse gold, if present, can be saved by amalgamation at some later stage of the manipulation.

Generally speaking, the number of commercial successes of the process in the United States is limited, although the number of tests and trials on a smaller or larger scale have been very numerous. The amount of ores suitable for the treatment is large, and the process is becoming more and more an established and acknowledged fact with the mining public. The miners of this country have long been looking for a process for the treatment of low-grade sulphurets which is cheaper in its application than chlorination. Under certain conditions the cyanide process meets the requirements, and will be found particularly valuable in remote places to which the freight expenses are high. The weight of chemicals used in chlorination amounts to about 5 per cent of the ore weight, but only to about 1 per cent in the case of cyanide treatment. By chlorination one ton of chemicals will treat about twenty tons of ore, whereas by cyanide one ton will treat 100 tons; moreover, the cyanide process does not require a special treatment of the ore for the extraction of the silver. No statistics in reference to the gold produced in the United States of America by the cyanide process could be obtained by the author.

## D. Mexico, Etc.

A MacArthur-Forrest company is introducing the process in that old mining country. No information, however, could be obtained from that company in reference to their results. In other parts of the world, attempts have been made, chiefly by the owners of the MacArthur-Forrest patents, to introduce the cyanide process. Ores from the republic of Colombia are reported as having been treated with success, and the introduction of the process into that country is now intended. Negotiations for the introduction of the process into the Straits Settlements, Borneo, the mining States of South America, and that great gold-producing country, Russia, are now pending.

I have in this paper been following the cyanide process in its workings and results through the chief places of its application. The process, like most metallurgical processes, has its weak points, which by continued investigations may be strengthened; its strong points are evident from the description of its successful application. Whatever may be the merit of the controversy on the subject of patent-rights in connection with this

process, there is no doubt that Messrs. MacArthur and Forrest deser the credit of having first, on a large scale, practically and successfull applied the cyanide process for working ores.

## SUMMARY AND CONCLUSIONS.

Generally speaking, the cyanide process is better suited for the trea ment of gold ores than of silver ores, one of the reasons of which may l found in the great variety of compounds in which the silver occurs, an a number of these offer difficulties in treatment. The required lor contact of cyanide, connected with the large consumption of reagen prevents the treatment of silver ores in some instances from being commercial success, even when chemically a high extraction is obtaine In reference to the general characteristics of ores which can be succes fully treated, it must be said that no definite specification can be give The question, Is an ore suited for cyanide treatment? can only be decid experimentally. Preliminary but exhaustive experiments on a limit scale should precede all operations on a large scale. Many of the pro lems of the process are of a chemical nature; many of its difficulties ar however, of a mechanical character, and experience and judgment mu guide in the selection of the plant to make the venture a financial su cess. One of the great mechanical difficulties has been, and is still, tl treatment of the slimes, by which is meant the very finest parts and tl clayey portion of ores and tailings. The final solution of the difficul in their treatment will probably be found in dry-crushing the ore an direct treatment of the crushed material with cyanide. In cases whe coarse gold is present, the amalgamation of such may be introduced some later periods of the manipulation. Practical experience with tl process extends over only a few years. It has been found to be we adapted for free-milling ores with finely divided gold, particularl so-called float gold, and has given great satisfaction with some pyrit ores. Even complex ores containing tellurides have been treated advantage. The process has found its most extensive application the Transvaal gold fields; although the average extraction is n high, it answers there better than any other process attempted f working the tailings. In other parts of the gold-producing world, i application is gaining way by degrees. Like all metallurgical p cesses, its success depends on the character of the ore and local c cumstances, and failures had to be recorded where these were sufficiently considered. It is certain that our knowledge is as incomplete, and there is still a large amount of ground for the metall gist and chemist to explore. We have yet particularly to learn h to extend the application of the process to more common use. A co parison of the cyanide process with other processes is a futile task; great merit of the treatment is that it comes to fill a want long felt—tl of treating low-grade ores and tailings in a simple and inexpensive w It is here that the process antagonizes no other methods, and sim takes its place in gold metallurgy as a new and powerful means to c quer nature. Wherever the process gives satisfactory results, it off great advantages: it does not require roasting furnaces; ores containi lead, zinc, or earthy carbonates, which cannot be worked to a profit chlorination, may be easily and profitably treated by it; as it does

require smelters, coal, and fluxes, it may be successfully used in remote situations, where smelting is absolutely impossible. One of its great advantages is, it does not require extra treatment for silver, invariably associated with gold in ore. I have generally spoken about gold only in this paper. The remarks referring to it and to its extraction apply with equal force to the silver associated with it. A well-constructed plant and efficient chemical and metallurgical supervision are, however, conditions always necessary to make its application a commercial success. The process is only in its infancy; many of the various and complex problems it has given rise to—such as the reduction in the consumption of cyanide and its regeneration—are still open questions. Its possibilities are great; chemical and mechanical improvements will enlarge the range of its application and usefulness. If it has not proved itself to be the metallurgical panacea that some enthusiasts expected, it has certainly during the four years of its technical application developed into a process of enormous economical importance, and one which justly may be considered a most valuable addition to gold metallurgy.

# APPENDIX.

# APPENDIX.

## UNITED STATES PATENT OFFICE.

JULIO H. REA, OF SYRACUSE, NEW YORK.

**IMPROVED MODE OF TREATING AURIFEROUS AND ARGENTIFEROUS ORES.**

Specification forming part of Letters Patent No. 61,866, dated February 5, 1867.

*To all whom it may concern:*

Be it known that I, Julio H. Rae, of Syracuse, in the county of Onondaga and State of New York, have invented a new and useful Improvement in Treating Auriferous and Argentiferous Ores; and I do hereby declare that the following is a full, clear, and exact description thereof, which will enable those skilled in the art to make and use the same, reference being had to the accompanying drawing, forming part of this specification, in which—

Figure 1 represents a transverse vertical central section of the apparatus which may be used in carrying out this invention.

Figure 2 is a plan or top view of the same.

Similar letters of reference in both views indicate corresponding parts.

This invention consists in treating auriferous and argentiferous ores with a current of electricity or galvanism for the purpose of separating the precious metals from the gangue. In connection with the electric current suitable liquids or chemical preparations, such, for instance, as cyanide of potassium, are used, in such a manner that by the combined action of the electricity and of the chemicals, the metal contained in the ore is first reduced to a state of solution and afterwards collected and deposited in a pure state, and that the precious metals can be extracted from the disintegrated rock or ore at a very small expense and with little trouble or loss of time.

In carrying out this process a jar, A, may be used such as shown in the drawing. This jar is made of glass or other suitable material (the size depending upon the electric battery to be used in connection therewith), and into said jar is placed the pulverized rock, filling the same half full or more. On the rock is poured the proper chemical preparation in a fluid state, such, for instance, as cyanide of potassium. Through the center of the jar passes a vertical shaft C, which terminates upon a metal plate, *a*, by preference of platina, which rests on the bottom of the jar, and to said shaft is attached a cage, B, of platina wire or other suitable material. This cage is made in the form of a truncated cone, its base extending close to the inner circumference of the shaft, or it may be made in any other suitable form or shape, a series of spirals, for instance, which will produce the same effect. On the shaft C is mounted a pulley, E, which may be connected with any suitable mechanism for the purpose of imparting a rotary motion to said shaft and the cage connected therewith, so that the contents of the jar will be agitated and each particle of the pulverized rock shall come in contact with the metal cage B and plate *a*. The shaft C is connected by a wire, *b*, with one, say the positive pole of the battery, thus converting the shaft, the cage, and the plate *a* into an electrode, and the other or negative pole of the battery connects by a wire, *c*, with a thin slip or coil, *d*, of copper or other suitable material, forming a base on which the precious metals are deposited. By the action of the electric current the action of the chemicals on the metals contained in the rock is materially faciliated and a perfect solution thereof is effected, and from this solution the precious metals are precipitated upon the base *d*, whence the same can be easily collected. By this process gold or silver can be extracted from rock in an absolutely pure state and with very little expense.

What I claim as new, and desire to secure by Letters Patent, is—

1. The within-described process of treating auriferous or argentiferous rock by exposing the same to the combined action of a current of electricity and of suitable solvents or chemicals, substantially such as herein specified, or any others which will produce the same effect.
2. Separating gold or silver from the rocks containing the same by the action or aid of electricity, substantially as described.
3. Using the agitator B as an electrode substantially as and for the purpose set forth.

JULIO H. RAE.

Witnesses:
W. HAUFF.
GEO. F. SOUTHERN.

J. H. RAE.

Treating Ores.

No. 61,866. Patented Feb. 5, 1867.

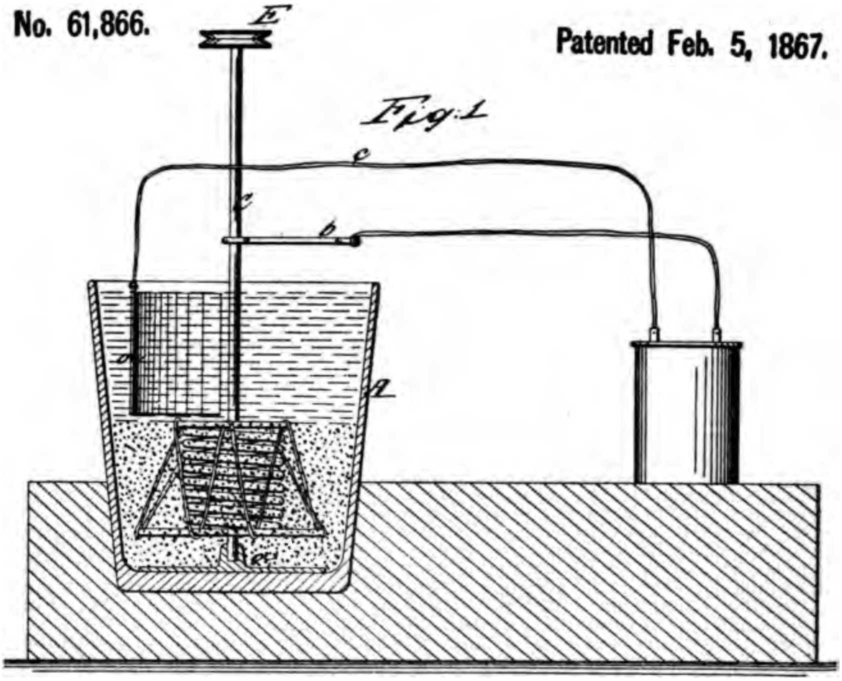
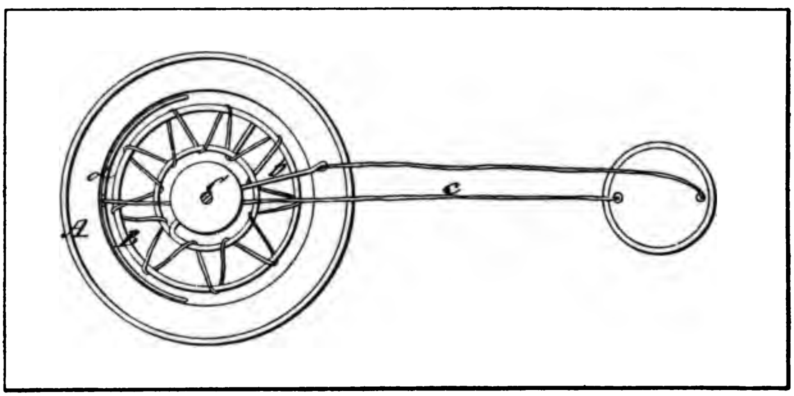



Fig. 2




Witnesses:
W. Hauff

Inventor
Julio H. Rae

# UNITED STATES PATENT OFFICE.

THOMAS C. CLARK, OF OAKLAND, ASSIGNOR TO JAMES STRATTON, OF SAME PLACE, AND RICHARD E. COADY, OF ALAMEDA, CALIFORNIA, ONE FOURTH TO EACH.

## EXTRACTING PRECIOUS METALS FROM ORES.

Specification forming part of Letters Patent No. 229,586, dated July 6, 1880.

(Application filed December 27, 1879.)

*To all whom it may concern:*

Be it known that I, Thomas C. Clark, of Oakland, county of Alameda, and State of California, have invented an Improvement in Extracting Precious Metals from ores; and I hereby declare the following to be a full, clear, and exact description thereof.

The object of my invention is to perform the disintegration and desulphurization of ores so as to bring the said ore into proper condition for easy pulverization and the precious metals contained therein into a suitable form for amalgamation by freeing them from the union and influence of baser metals. In order to accomplish this object the ore is crushed into pieces about the size of ordinary Indian corn. That portion containing sulphurets generally becomes finer, since it is more friable. The object of crushing it to this size is to prevent loss of gold and to facilitate washing operations.

The ore, after being crushed as described, is placed in an ordinary roasting-furnace. After being roasted for a suitable length of time the heat is raised, so the sulphur will burn freely, after which the heat is let down again, a free supply of oxygen being furnished during the whole process of roasting.

After the ore has become dead and lies like sand in the furnace, and no more scintillation is apparent, it is heated up to a good red heat, but not made too hot.

In a suitable receptacle beside the furnace I form a cold bath, into which the ore is drawn while in its heated condition fresh from the furnace. This bath is formed of a solution of salt, prussiate of potash, and caustic soda or caustic potash.

For one ton of gold ore containing five per cent or less of sulphurets, I form my bath in about the following proportions: I take about thirty gallons of cold water, to which common salt is added until a saturated solution is formed. I then dissolve one pound of prussiate of potash in water and pour it into the solution, and also dissolve one pound of caustic soda in water and add it to the solution. The bath then contains chloride of sodium, prussiate of potash, and caustic soda. For the latter caustic potash may be substituted with a like result.

The red-hot ore being drawn into the cold solution described, a complete desulphurization is effected, as well as a disintegration.

Where there is a higher percentage of sulphur in the ore, additional quantities of the prussiate of potash and caustic soda are added, the proportions of the solution being thus altered to suit the requirements of the class of ore under treatment. The proportions may also be modified for ore of different character.

I am aware that ore has frequently been roasted and dumped while red hot into cold water or into cold solutions, and I therefore do not claim, broadly, such process; but

What I do claim as new, and desire to secure by Letters Patent, is—

The process of disintegrating and desulphurizing ores and freeing the precious metals therein contained, consisting in first roasting the ore to a red heat, and while in that condition placing it in a cold bath composed of a solution of salt, prussiate of potash, and caustic soda or caustic potash, in about the proportions named, substantially as herein described.

In witness whereof I have hereunto set my hand.

THOMAS C. CLARK.

Witnesses:
CHAS. G. YALE.
S. H. NOURSE.

# UNITED STATES PATENT OFFICE.

HIRAM W. FAUCETT, OF ST. LOUIS, MISSOURI.

## PROCESS OF TREATING ORE.

Specification forming part of Letters Patent No. 236,424, dated January 11, 1881.

(Application filed July 13, 1880. No specimens.)

*To all whom it may concern:*

Be it known that I, Hiram W. Faucett, a citizen of the United States, residing at St. Louis, in the county of St. Louis and State of Missouri, have invented new and useful Improvements in Process of Treating Ores, of which the following is a specification.

The object of my invention is to treat all refractory ores containing gold and silver for the purpose of separating such metals from the ore.

It is well known that the chemical nature of all refractory ores is more or less of a silicious character, and by desulphurizing or roasting such ores they are rendered porous. Taking advantage of this fact, I subject the silicious ore to a proper chemical bath under pressure, which effectually disintegrates the ore by decomposing or destroying the silica therein, which silica is the chemical agent in the ore which holds or locks the ore together in a compact mass.

To this end my invention consists, broadly, in subjecting hot crushed ores to the action of disintegrating chemicals in solution while under pressure, the pressure being effected by the steam generated by the contact of the hot ore and the chemical solution in a closed vessel.

In carrying out this process I take the ore, crushed as ordinarily for stamp-mill or smelter, and heat the same in any suitable furnace to a sufficient degree and for a proper time to desulphurize it. I then draw the ore, while at red heat, into an iron retort of proper strength to withstand the proposed pressure, and provided with a steam-tight door, and into this retort, through a suitable aperture, after closing the door, I introduce the chemicals in a state of solution, the steam generated creating such pressure within the retort that the chemicals are forced into the silica or rock of the ore, thoroughly disintegrating the same and freeing the metals therefrom, so that the latter are rendered susceptible of ready amalgamation. The process will be facilitated by agitating the retort.

I use different chemicals, according to the different kinds of ore to be treated, and the quantity required depends upon the quantity of silica or other refractory substances to be decomposed to effect a thorough disintegration of the ore.

For the treatment of most of the refractory ores I use chloride of sodium as a base, and in connection therewith nitrate of potash, cyanide of sodium, and about equal parts of sulphate of protoxide of iron and sulphate of copper, the proportions being about as follows, viz.: chloride of sodium, from thirty to forty pounds to the ton; nitrate of potassium, from one to two pounds to the ton; cyanide of sodium, from two to four pounds to the ton; sulphate of protoxide of iron, from one to two pounds to the ton; sulphate of copper, from one to two pounds to the ton. These chemicals are to be dissolved in boiling or hot water of sufficient quantity to cover the ore in the retort. If the ores to be treated are unusually hard or refractory, I add to the above one to two pounds of hydrofluoric acid, or one to two pounds of fluoride of potassium or fluoride of sodium, according to the character of the ore. After the ore has been agitated in the retort a proper time—say from ten to fifteen minutes—under from fifty to one hundred pounds pressure to the square inch, it may be removed while hot to the pulverizer, then passed through any convenient and desired amalgamating process.

It should be here stated that while the ore is in the chemical bath, the latter acts to disintegrate the ore by decomposing or destroying the silica therein, and the ore is thoroughly impregnated with the chemicals, thereby effectually disengaging the particles of metal from the silica, which, if not disengaged, will not amalgamate with the quicksilver in the amalgamating-machines.

To facilitate the carrying out of the process, I prefer to use a cylindrical retort mounted on axial trunnions, in order that it may be rotated for the purpose of agitating its contents. The door of the retort should be in its side, and at each end there should be a projecting coupling-nipple provided with a cut-off cock, to which may be connected pipes, one of which leads from the top, and the other from the bottom, of an elevated steam-tight receiver provided with safety-valve, both pipes being provided with suitable cocks. The chemical solution is then placed in the elevated cylinder, and, after the retort has received its charge of heated ore and been closed, the pipes from the receiver are connected to the coupling-nipples and the cocks all opened. The solution will flow from the bottom of the tank to the retort, and the steam generated in the latter will flow to the top of the cylinder, creating a pressure therein which will force the solution rapidly into the retort. After the pressure has decreased, the cocks may then be

closed, the pipes disconnected from the retort, and the latter rotated for the purpose of agitating its contents.

The apparatus thus partially described will form the subject of a separate application for Letters Patent.

I do not confine myself to the chemicals or quantities thereof herein enumerated, as they may be varied as required by the character of the ores to be treated.

I am aware that crushed ores have been subjected to the action of chlorine gas under pressure, and that unroasted pulverized ores have been treated with chemical solutions in a closed vessel under pressure created by the injection of steam, and also that hot roasted ores have been treated by placing cold chemical solutions in contact therewith in the open air, or not under pressure; and I do not claim any of such modes of treatment.

What I claim is—

1. The process herein described for separating metals from ores, the same consisting in subjecting hot crushed ores to the action of disintegrating chemicals in solution under pressure, the pressure being effected by the steam generated by the contact of the hot ores with the chemical solution in a closed vessel, substantially as specified.

2. The process herein described for treating refractory ores for disengaging the precious metals therefrom, the same consisting in subjecting hot crushed ores to the action of a solution of chloride of sodium, nitrate of potash, cyanide of sodium, sulphate of protoxide of iron, and sulphate of copper, under pressure, with or without admixture of hydrofluoric acid, fluoride of potassium, or fluoride of sodium, the pressure being effected by the steam generated by the contact of the solution with the hot ore, substantially as set forth.

In testimony whereof I have hereunto set my hand in the presence of two subscribing witnesses.

HIRAM W. FAUCETT.

Witnesses:
ROBT. HARBISON,
JNO. C. ORRICK.

## UNITED STATES PATENT OFFICE.

JOHN F. SANDERS, OF OGDEN, UTAH TERRITORY.

### COMPOSITION FOR DISSOLVING THE COATING OF GOLD IN ORE.

Specification forming part of Letters Patent No. 244,080, dated July 12, 1881.

(Application filed April 16, 1881. No specimens.)

*To all whom it may concern:*

Be it known that I, John F. Sanders, of Ogden, in the county of Weber and Territory of Utah, have invented an improved Composition for Dissolving the Coating of Gold in Ore, of which the following is a specification:

The coatings that envelop gold in the ore, and that consist usually of various metallic oxides and of silver, have thus far been difficult to remove, except under the influence of extreme heat, which it is not possible at all places to apply, or by the waste of much valuable time. I have found that a mixture of cyanide of potassium and phosphoric acid, in about the proportions hereinafter mentioned, constitutes a powerful solvent for these coatings of gold ore.

I use in my mixture about sixteen parts of cyanide of potassium to one part glacial phosphoric acid. These two ingredients I mix shortly before the mixture is to be used. The mixture I place into the vessel that contains the covered ore. This vessel preferably is a rotating barrel made of iron or other proper material, and the composition above named is added with sufficient water to form a thick pulp with the raw gravel. The proportions of my improved mixture to the ore will vary, of course, with the varying thickness of covering of the gold. They will, however, be readily ascertained by testing with samples of the ore to be treated. The barrel is rotated or agitated in suitable manner for from fifteen to sixty minutes, as may be required. After agitation the mixture above mentioned will be found, on investigation, to have dissolved the oxides and the sulphurous coatings of the ore, and the agitation of the barrel or vessel removes the dissolved impurities, leaving the gold free and exposed, and permitting it to be amalgamated by the addition of quicksilver, in the usual manner.

The amalgam may be separated from the impurities which have joined with the improved mixture in the manner in which amalgams are usually separated from impurities.

I am aware that cyanides have already been used in the extraction of gold; also, that gold-bearing ores have been disintegrated in the presence of heat by various chemicals. This I do not claim. By using phosphoric acid in the presence of cyanide of potassium I am enabled to dissolve the impurities in a raw state and with great rapidity.

I claim—

The composition of cyanide of potassium and phosphoric acid, in about the proportions mentioned, for the purpose of dissolving the impure coatings of gold, substantially as specified.

JOHN F. SANDERS.

Witnesses:
WILLY G. E. SCHULTZ,
WILLIAM H. C. SMITH.

# UNITED STATES PATENT OFFICE.

JEROME W. SIMPSON, OF NEWARK, NEW JERSEY.

## PROCESS OF EXTRACTING GOLD, SILVER, AND COPPER FROM THEIR ORES.

Specification forming part of Letters Patent No. 323,222, dated July 28, 1885.

(Application filed October 20, 1884. No specimens.)

*To all whom it may concern:*

Be it known that I, Jerome W. Simpson, a citizen of the United States, residing at Newark, in the county of Essex, and State of New Jersey, have invented certain new and useful improvements in processes of extracting gold, silver, and copper from their ores; and I do hereby declare the following to be a full, clear, and exact description of the invention, such as will enable others skilled in the art to which it appertains to make and use the same.

The object of this invention is to extract certain metals from their ores more effectually and at a reduced cost; and it consists in the processes hereinafter set forth, and finally embodied in the clauses of the claims.

To carry my invention into effect, I first grind or crush the ore containing the metal to be extracted to a powder of more or less fineness. This powder is then treated with certain salts in solution adapted to combine chemically with the metal in said ore and form therewith a soluble salt. After thorough agitation to mix the solution with the ore, the mixture is allowed to stand until the solid matter is settled and the solution has become clear. I then suspend a piece or plate of zinc therein, which causes the metal dissolved in the salt solution to be precipitated thereon, from which it can be removed by scraping or by dissolving the zinc in sulphuric or hydrochloric acid. The precipitated metal may then be melted into a button.

The salt solution I use for dissolving the metal from the ore is composed of one pound of cyanide of potassium, one ounce carbonate of ammonia, one half ounce chloride of sodium, and sixteen quarts of water, or other quantities in about the same proportions.

This solution is particularly adapted for ores containing gold, silver, and copper in the form of sulphurets.

For an ore containing gold and copper only I use cyanide of potassium and carbonate of ammonia about in the proportions named.

For ores rich in silver I employ a proportionately larger quantity of chloride of sodium.

I am aware that cyanide of potassium, when used in connection with an electric current, has been used for dissolving metal, and also that zinc has been employed as a precipitant, and the use of these I do not wish to be understood as claiming, broadly.

I am also aware that carbonate of ammonia has been employed for dissolving such metals as are soluble in a solution thereof, and the use of this I do not claim; but

What I claim as new is—

1. The process of separating gold and silver from their ores, which consists in subjecting the ore to the action of a solution of cyanide of potassium and carbonate of ammonia, and subsequently precipitating the dissolved metal, substantially as set forth.

2. The process of separating metals from their ores, to wit: subjecting the ore to the action of a solution of cyanide of potassium, carbonate of ammonia, and chloride of sodium, and subsequently precipitating the dissolved metals.

In testimony that I claim the foregoing, I have hereunto set my hand this 15th day of October, 1884.

JEROME W. SIMPSON.

Witnesses:
OLIVER DRAKE.
CHARLES H. PELL.

[Fifth Edition.]

No. 14,174. A. D. 1887.

Date of application, 19th Oct., 1887; complete specification left, 26th July, 1888—Accepted 10th Aug., 1888.

PROVISIONAL SPECIFICATION.

**IMPROVEMENTS IN OBTAINING GOLD AND SILVER FROM ORES AND OTHER COMPOUNDS.**

We, John Stewart MacArthur, Analytical Chemist, of 15 Princes Street, Pollokshields, in the County of Renfrew, North Britain, Robert Wardrop Forrest, M.D., and William Forrest, M.B., both of 319 Crown Street, Glasgow, in the County of Lanark, North Britain, do hereby declare the nature of this invention to be as follows:

This invention has principally for its object the obtaining of gold from its ores or other compounds, but it is also applicable for obtaining silver from its ores or compounds; and it comprises an improved process, which, whilst applicable to ores or compounds generally, is effectual with ores and compounds from which gold or silver have hitherto not been easily obtainable.

In carrying out the invention the ore or other compound in a powdered state is treated with a solution containing cyanogen or a cyanide (such as the cyanides of potassium, sodium, or ammonium), or other substance or compound containing or yielding cyanogen, till all or nearly all of the gold and the silver are dissolved; the operation being conducted in a wooden vessel or a vessel made of or lined with a material not acted on to any considerable extent by the solution or substances contained therein. The solution is then drawn off and the metal or metals are recovered by any suitable process, and the cyanogen, cyanide, or substance containing or yielding cyanogen may be regenerated. The cyanogen or substance containing or yielding cyanogen may be used as such, or such materials may be taken as will by mutual action form cyanogen or substances containing or yielding same.

Under certain circumstances it may be found desirable to conduct the operation under pressure, in which case a closed vessel must be employed, and in any case, if found advisable, such operation may be carried on under varying conditions of temperature.

Dated this 19th day of October, 1887.

ALLISON BROS.,
Agents for the Applicants.

COMPLETE SPECIFICATION.

**Improvements in Obtaining Gold and Silver from Ores and Other Compounds.**

We, John Stewart MacArthur, Analytical Chemist, of 15 Princes Street, Pollokshields, in the County of Renfrew, North Britain, Robert Wardrop Forrest, M.D., and William Forrest, M.B., both of 319 Crown Street, Glasgow, in the County of Lanark, North Britain, do hereby declare the nature of this invention, and in what manner the same is to be performed, to be particularly described and ascertained in and by the following statement, that is to say:

This invention has principally for its object the obtaining of gold from its ores or other compounds, but it is also applicable for obtaining silver from its ores or compounds; and it comprises an improved process, which, whilst applicable to ores or compounds generally, is effectual with ores and compounds from which gold or silver have hitherto not been easily obtainable because of the presence of various other metals or their compounds, or because of the physical or chemical condition of the gold or silver in the ores or compounds.

In carrying out the invention the ore or other compound in a powdered state is treated with a solution containing cyanogen or cyanide (such as cyanide of potassium, or of sodium, or of calcium), or other substance or compound containing or yielding cyanogen. In practice we find the best results are obtained with a very dilute solution, or a solution containing or yielding an extremely small quantity of cyanogen or a cyanide, such dilute solution having a selective action such as to dissolve the gold or silver in preference to the baser metals. In preparing the solution we proportion the cyanogen to the quantity of gold or silver or gold and silver estimated by assay or otherwise to be in the ore or compound under treatment, the quantity of a cyanide or cyanogen-yielding substance or compound being reckoned according to its cyanogen. We mix the powdered ore, or compound, with the solution in a vessel made of or lined with wood or any other convenient material not appreciably acted on by the solution. The process is

expedited by stirring the mixture of ore and solution intermittently, or continuously, for which purpose any convenient mechanical agitator may be fitted to the vessel. When all or nearly all the gold or silver is dissolved the solution is drawn off from the ore or undissolved residue, and is treated in any suitable known way, as for example with zinc, for recovering the gold and silver. The residuary cyanogen compounds may also be treated by known means for regeneration or reconversion into a condition in which they can be used for treating fresh charges of ores or compounds.

Any cyanide soluble in water may be used, such as ammonium, barium, calcium, potassium, or sodium cyanide, or a mixture of any two or more of them, or any mixture of materials may be taken which will, by mutual action, form cyanogen, or a substance or substances containing or yielding cyanogen.

In dealing with ores or compounds containing, per ton, twenty ounces or less of gold or silver, or gold and silver, we generally use a quantity of cyanide, the cyanogen of which is equal in weight to from one to four parts in every thousand parts of the ore or compound, and we dissolve the cyanide in a quantity of water of about half the weight of the ore. In the case of richer ores or compounds, whilst increasing the quantity of cyanide to suit the greater quantity of gold or silver, we also increase the quantity of water so as to keep the solution dilute. In using free cyanogen, the cyanogen obtained as a gas in any well known way is led into water to form the solution to be used in our process; or any suitable known mode of setting cyanogen free in solution may be employed.

In some circumstances it may be found desirable to conduct the operation under pressure in a closed vessel; and a higher than the ordinary temperature may be used if found desirable.

Having now particularly described and ascertained the nature of our said invention and in what manner the same is to be performed, we declare that what we claim is—

1. The process of obtaining gold and silver from ores and other compounds, consisting in dissolving them out by treating the powdered ore or compound with a solution containing cyanogen or a cyanide or cyanogen-yielding substance, substantially as hereinbefore described.

2. The process of obtaining gold and silver from ores and other compounds, consisting in dissolving them out by treating the powdered ore or compound with a dilute solution containing a quantity of cyanogen or a cyanide or cyanogen-yielding substance, the cyanogen of which is proportioned to the gold or silver or gold and silver, substantially as hereinbefore described.

Dated this 16th day of July, 1888.

ALLISON BROS.,
Agents for the Applicants.

## UNITED STATES PATENT OFFICE.

JOHN STEWART MACARTHUR, OF POLLOKSHIELDS, COUNTY OF RENFREW, AND ROBERT W. FORREST AND WILLIAM FORREST, OF GLASGOW, COUNTY OF LANARK, SCOTLAND.

### PROCESS OF OBTAINING GOLD AND SILVER FROM ORES.

Specification forming part of Letters Patent No. 403,202, dated May 14, 1889.

(Application filed November 9, 1887. Serial No. 254,699. (No specimens.) Patented in England, October 19, 1887, No. 14,174; in Cape of Good Hope, January 7, 1888, No. 6-101; in Victoria, January 19, 1888, No. 5,572; in New South Wales, January 21, 1888, No. 453; in South Australia, January 23, 1888, No. 948; in Tasmania, January 24, 1888, No. 529; in New Zealand, February 1, 1888, No. 2,775; in Canada, February 6, 1888, No. 28,471; in France, April 6, 1888, No. 189,808; in Belgium, July 24, 1888, No. 82,673; in Brazil, August 23, 1888, No. 619; in Portugal, August 30, 1888, No. 1,272; in Italy, September 30, 1888, No. 23,852, and in Spain, October 2, 1888, No. 8,538.)

*To all whom it may concern:*

Be it known that we, John Stewart MacArthur, a subject of the Queen of Great Britain, residing at 15 Princes Street, Pollokshields, in the County of Renfrew, Scotland, and Robert Wardrop Forrest and William Forrest, both subjects of the Queen of Great Britain, residing at 319 Crown Street, Glasgow, in the County of Lanark, Scotland, have invented certain new and useful Improvements in Processes of Obtaining Gold and Silver from Ores (for which we have obtained patents in the following countries: Great Britain, No. 14,174, dated October 19, 1887; Cape of Good Hope, No. 6-101, dated January 7, 1888; Victoria, No. 5,572, dated January 19, 1888; New South Wales, No. 453, dated January 21, 1888; South Australia, No. 948, dated January 23, 1888; Tasmania, No. 529, dated January 24, 1888; New Zealand, No. 2,775, dated February 1, 1888; Canada, No. 28,471, dated February 6, 1888; France, No. 189,808, dated April 6, 1888; Belgium, No. 82,673, dated July 24, 1888; Brazil, No. 619, dated August 23, 1888; Portugal, No. 1,272, dated August 30, 1888; Italy, No. 23,852, dated September 30, 1888, and Spain, No. 8,538, dated October 2, 1888); and we do hereby declare that the following is a full, clear, and exact description of the invention, which will enable others skilled in the art to which it appertains to make and use the same.

This invention has principally for its object the obtaining of gold from ores; but it is also applicable for obtaining silver from ores containing it, whether with or without gold, and it comprises an improved process which, while applicable to auriferous and argentiferous ores generally, is advantageously and economically effective with refractory ores, or ores from which gold and silver have not been satisfactorily or profitably obtainable by the amalgamating or other processes hitherto employed, such as ores containing sulphides, arsenides, tellurides, and compounds of base metals generally, and ores from which the gold has not been easily or completely separable on account of its existing in the ores in a state of extremely fine division.

The invention consists in subjecting the auriferous or argentiferous ores to the action of a solution containing a small quantity of a cyanide, as hereinafter set forth, without any other chemically-active agent, such quantity of cyanide being reckoned according to its cyanogen, and the cyanogen being proportioned to the quantity of gold or silver, or gold and silver, estimated by assay or otherwise to be in the ores under treatment. By treating the ores with the dilute and simple solution of a cyanide the gold or silver is, or the gold and silver are, obtained in solution, while any base metals in the ores are left undissolved, except to a practically inappreciable extent, whereas when a cyanide is used in combination with an electric current or in conjunction with another chemically-active agent—such as carbonate of ammonium, or chloride of sodium, or phosphoric acid—or when the solution contains too much cyanide, not only is there a greater expenditure of chemicals in the first instance, but the base metals are dissolved to a large extent along with the gold or silver, and for their subsequent separation involve extra expense, which is saved by our process.

In practically carrying out our invention we take the ore in a powdered state and mix with it the solution of a cyanide in a vessel made of or lined with any material not appreciably acted on by the solution. We employ a vessel made of or lined with wood; but it may be made of or lined with any ordinary inert material—such as stone, brick, slate, rubber, gutta-percha, cement, glass, earthenware, iron (plain, tinned, or enameled), or lead. The process is expedited by stirring or triturating the mixture of ore and solution intermittently or continuously, for which purpose any convenient mechanical agitator may be fitted to the vessel. A pan-mill with edge runners or other known triturating device may be advantageously used. The solution is allowed to act on the ore until the gold or silver is all or nearly all dissolved, and the solution is then drawn off from the ore or undissolved residue.

Any cyanide soluble in water may be used—such as ammonium, barium, calcium, potassium, or sodium cyanide, or a mixture of any two or more of them. We regulate the quantity of cyanide so that its cyanogen will be in proportion to the quantity of

gold or silver or gold and silver in the charge of ore; but in all cases we dissolve it in sufficient water to keep the solution extremely dilute, because it is when the solution is dilute that it has a selective action such as to dissolve the gold or silver in preference to the baser metals.

In dealing with ores containing per ton twenty ounces or less of gold or silver or gold and silver, we find it most advantageous to use a quantity of cyanide the cyanogen of which is equal in weight to from one to four parts for every thousand parts of the ore, and we dissolve the cyanide in a quantity of water of about half the weight of the ore. We generally use a solution containing two parts of cyanogen for every thousand parts of the ore. In the case of richer ores, while increasing the quantity of cyanide to suit the greater quantity of gold or silver, we also increase the quantity of water so as to keep the solution dilute. In other words, the cyanide solution should contain from two to eight parts, by weight, of cyanogen to one thousand parts of water, and the quantity of the solution used should be determined by the richness of the ore. After the solution has been decanted or separated from the undissolved residues the gold and silver may be obtained from it in any convenient known way—such as evaporating the solution to dryness and fusing the resulting saline residue, or by treating the solution with sodium amalgam.

Having fully described our invention, what we desire to claim and secure by Letters Patent is—

The process of separating precious metal from ore containing base metal, which process consists in subjecting the powdered ore to the action of a cyanide solution containing cyanogen in the proportion not exceeding eight parts of cyanogen to one thousand parts of water.

JOHN STEWART MACARTHUR.
ROBT. W. FORREST.
W. FORREST.

Witnesses:
ROBT. DUNLOP,
WILLIAM BRUNTON,
Law clerks, both of 160 West George Street, Glasgow.

# UNITED STATES PATENT OFFICE.

JOHN STEWART MACARTHUR, OF POLLOKSHIELDS, COUNTY OF RENFREW, SCOTLAND.

## METALLURGICAL FILTER.

Specification forming part of Letters Patent No. 418,138, dated December 24, 1889.

(Application filed November 13, 1889. Serial No. 330,195. No model.)

*To all whom it may concern:*

Be it known that I, John Stewart MacArthur, a subject of the Queen of Great Britain, residing at Pollokshields, in the county of Renfrew, Scotland, have invented a new and useful Improvement in Metallurgical Filters, of which the following is a specification:

This invention relates to a filter for precipitating and separating precious metals from solutions containing them—such, for instance, as chlorides, bromides, theosulphates (sometimes called "hyposulphites"), or sulphates obtained in the well-known Plattner, von Patera, Russell, Ziervogel, and Augustine extracting processes.

The object of the invention is to provide a filter having a large active surface for the metals in solution.

In the accompanying drawings, Figure 1 is a sectional elevation of a series of these improved filters. Fig. 2 is a longitudinal vertical section of a filtering apparatus comprising two of these improved filters constructed in modified form. Fig. 3 represents a zinc filiform sponge, constituting the principal feature of this improved filter, the filaments of the sponge being represented on an enlarged scale.

Similar numerals of reference indicate corresponding parts in the different figures.

This improved filter comprises a containing-vessel 10 and a zinc sponge 11, disposed therein. The zinc sponge is preferably supported on a perforated false bottom 12, disposed within said vessel near the bottom proper thereof. The vessel is provided with an inlet-tube 13 and an outlet-tube 14, the inlet-tube being preferably disposed near the bottom of the vessel and the outlet-tube near the top thereof, each of said tubes being provided with a coupling-nut 15 when the vessels are used in series.

A number of these filters are preferably arranged in series, as represented in Fig. 1, from six to ten being ordinarily employed. When so arranged, the filters are connected by pipes 16, which extend from the outlet near the top of one vessel to the inlet near the bottom of the adjacent vessel. A reservoir or tank 17 for containing the solution holding the precious metals is disposed adjacent to the first filter of the series and elevated a sufficient distance to secure a proper flow of the liquid through the filters. This tank is provided with an outlet-tube 18 near its bottom, said tube being provided with a stop-cock 19, and connected by pipe 20 with the inlet-tube of the first filter of the series. The zinc sponge, which constitutes the filtering material and precipitant, is preferably composed of fine threads or filaments of zinc interlocked together. The zinc threads from which the sponge is formed are cut by a turning tool from a series of zinc disks held between lathe-centers and turned; or the zinc sponge may be formed by passing molten zinc, at a temperature just above the melting-point, through a fine sieve and allowing it to fall into water. This improved zinc sponge presents a very large contact-surface for the action of the solution, and it does not become easily choked. Each containing-vessel may be provided with a vertical partition or partitions 21, as illustrated in Fig. 2, whereby the vessel is divided into two or more compartments or filtering-chambers. These partitions extend to a point near the bottom or top of the vessel, as the case may be, or they are provided with holes near the top or near the bottom of the vessel. In case the vessel has three or more filtering-chambers, the partitions are provided with communicating-openings, disposed alternately near the bottom and top of the vessel, whereby the passage of the solution is downward through one of the filtering-chambers, upward through the adjoining filtering-chamber, and downward again through the third filtering-chamber, and so on.

In the use of this improved filtering apparatus the solution containing the precious metal is placed in the tank 17 and the cock 19 is opened. In case a series of separate filters is employed, as represented in Fig. 1, the solution passes from the tank through the pipe 20 and into the first filter of the series, near the bottom thereof, beneath the false bottom 12, thence upward through the zinc sponge within the filter, thence outward near the top of the first filter, thence through the connecting-pipe to the next filter of the series, where it again enters near the bottom and passes upward through the zinc sponge to the top of the second filter of the series, and so on. The metal which is not precipitated by the first filter is caught in the zinc sponge of the succeeding filters of the series.

In case filters having a number of compartments are employed, the solution is perferably admitted to the first compartment at the top thereof, and passes down through the zinc sponge contained in said compartment to near the bottom thereof, and thence passes into the second compartment and upward through the zinc sponge therein contained to near the top of said compartment, and thence downward through the next

(No Model.)

# J. S. MACARTHUR.

## METALLURGICAL FILTER.

No. 418,138. Patented Dec. 24, 1889.

Fig 1.

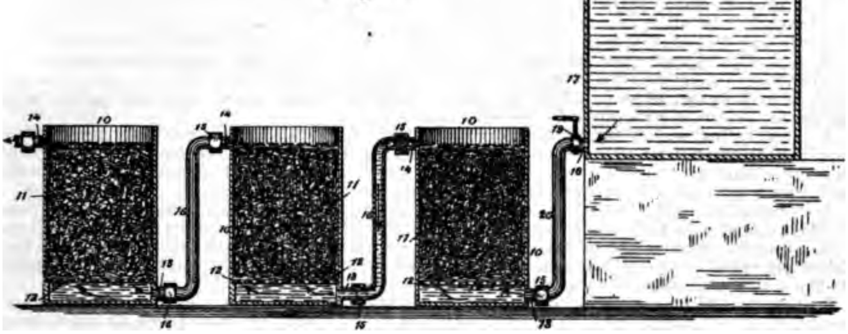


Fig 2.

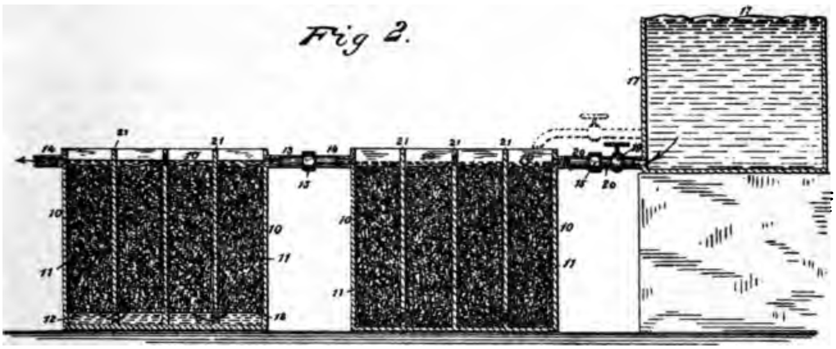


Fig 3.

WITNESSES.

Harry King

C. A. Weed.

INVENTOR

John Stewart MacArthur

By F. C. Somes,

Attorney

compartment, and so on through the several compartments of the compound filter, and thence to the next compound filter of the series and through its several compartments. The precious metal may be separated from the zinc sponge by distillation, or the zinc sponge containing the precious metal may be placed in a suitable sieve and subjected to a screening operation, preferably under water. In this operation the greater part of the precious metal will pass through the sieve and the greater part of zinc sponge will remain therein.

I claim as my invention—

1. A metallurgical filter for separating a precious metal from a solution containing said metal, consisting of a vessel provided with inlet and outlet openings, and a zinc sponge disposed in said vessel between said openings, substantially as described.

2. A metallurgical filter for separating a precious metal from a solution containing said metal, consisting of a vessel provided with inlet and outlet openings and a filiform zinc sponge disposed in said vessel between said openings, substantially as described.

3. A metallurgical filter for separating a precious metal from a solution containing said metal, consisting of a vessel provided with a perforated false bottom, a zinc sponge within said vessel above said false bottom, and inlet and outlet openings above and below said sponge, substantially as described.

4. A metallurgical filter for separating a precious metal from a solution containing said metal, consisting of a vessel provided with a perforated false bottom, a filiform zinc sponge within said vessel above said false bottom, and inlet and outlet openings above and below said filiform sponge, substantially as described.

5. A metallurgical filtering apparatus for separating a precious metal from a solution containing said metal, consisting of a series of vessels, a zinc sponge in each of said vessels, pipes connecting the outlet-tube of one vessel of the series with the inlet-tube of the adjacent vessel of the series, and a reservoir for supplying the solution to the first vessel of the series, substantially as described.

6. A metallurgical filtering apparatus for separating a precious metal from a solution containing said metal, consisting of a series of vessels, each of which has an inlet-tube near its bottom, an outlet-tube near its top, and a perforated false bottom above the inlet-tube, zinc sponges disposed in the several vessels, pipes connecting the inlet and outlet-tubes of the several vessels, and a reservoir for supplying the solution to the first vessel of the series, substantially as described.

7. A metallurgical filter for separating a precious metal from a solution containing said metal, consisting of a vessel provided with a partition dividing said vessel into a plurality of filtering-chambers, said partition being provided with openings near one end, and zinc sponges disposed in said compartments, substantially as described.

JOHN STEWART MACARTHUR.

Witnesses:
F. C. SOMES.
GORDON WILSON, JR.

# UNITED STATES PATENT OFFICE.

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JOHN STEWART MACARTHUR, OF POLLOKSHIELDS, COUNTY OF RENFREW, AND ROBERT WARDROP FORREST AND WILLIAM FORREST, OF GLASGOW, COUNTY OF LANARK, ASSIGNORS TO THE CASSEL GOLD EXTRACTING COMPANY (LIMITED) OF GLASGOW, SCOTLAND.

## PROCESS OF SEPARATING GOLD AND SILVER FROM ORE.

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Specification forming part of Letters Patent No. 418,137, dated December 24, 1889.

(Application filed April 4, 1889. Serial No. 305,998. [Specimens.] Patented in Natal, September 11, 1888, No. 32; in New South Wales, September 27, 1888, No. 965, and in Tasmania, September 29, 1888, No. 609.)

*To all whom it may concern:*

Be it known that we, John Stewart MacArthur, residing at Pollokshields, in the county of Renfrew, and Robert Wardrop Forrest and William Forrest, both residing at Glasgow, in the county of Lanark, Scotland, all subjects of the Queen of Great Britain, have invented certain new and useful improvements in the process of separating gold and silver from ores (for which we have received Letters Patent in Natal, No. 32, dated September 11, 1888; New South Wales, No. 965, dated September 27, 1888, and Tasmania, No. 609, dated September 29, 1888); and we do hereby declare that the following is a full, clear, and exact description of the invention, which will enable others skilled in the art to which it appertains to make and use the same.

This invention relates to an improvement in the process of separating precious metals from ores described in Letters Patent of the United States, No. 403,202, granted to us May 14, 1889. In that process a cyanide is used as the separating agent, and it has been found that ores containing pyrites or sulphurets which have been exposed to the weather and become partially oxidized absorb a comparatively large quantity of the cyanide.

The object of this invention is to economize the process by preventing the absorption of the cyanide.

The invention consists in separating precious metals from ores by first neutralizing the ore by the addition of an alkali or alkaline earth and then leaching such prepared charge with a cyanide solution.

In carrying out the first or preparatory step of this improved process, we take ore containing iron pyrites or other compound which has become partially oxidized by exposure to the weather and mix with it, when in a powdered state, a sufficient quantity of potash, lime, or other alkali or alkaline earth, to neutralize the salts of iron or other objectionable ingredients formed by the partial oxidation.

The quantity of alkali or alkaline earth to be employed will depend upon the nature of the ore, and must be determined by first taking a test quantity of the particular ore to be treated and adding the alkali or alkaline earth thereto until the alkali ceases to be absorbed. When this condition is reached the liquid will cause red litmus-paper to turn blue. The proportion of the alkali or alkaline earth so absorbed will indicate the proper proportion thereof to be added to the bulk of the ore to be treated. In case lime is employed, 1 per cent of the alkali to 99 per cent of ore will generally be found sufficient. After this preparatory treatment the ore (which may consist of tailings or residues from other processes or operations) is treated with the cyanide solution by being agitated therewith or by being ground therewith in a pan-mill or other suitable grinding-mill; or, as we find preferable in the case of some ores, the cyanide solution may be made to percolate through said ores one or more times until all or nearly all the precious metals are dissolved. For this percolation very simple tanks, vats, or vessels may be used, such vessels being provided with permeable false bottoms or any suitable filtering apparatus. The cyanide solution containing the gold or silver is next made to pass through a sponge of zinc, whereby said metal is precipitated from the solution and retained in the sponge. The zinc sponge is preferably composed of fine threads or filaments of zinc. These zinc threads are formed in shavings cut by a turning tool from a series of zinc disks held in a lathe; or the sponge may be formed by passing molten zinc at a temperature just above the melting point through a fine sieve and allowing it to fall into the water. The sponge thus formed presents a very large contact surface for the solution, and it does not become easily choked.

The precious metals may be separated from the zinc sponge by distillation; or the zinc sponge containing the precious metal may be placed in a suitable sieve and subjected to a screening operation, preferably under water. In this operation the greater part of the precious metal will pass through the sieve and the greater part of the zinc sponge will remain therein.

We claim as our invention—

1. The process of separating precious metals from an ore, which consists in neutralizing the ore by the addition of an alkali or alkaline earth, and then leaching the neutralized ore with a cyanide solution.

2. The process of separating precious metal from an ore, which consists in neutralizing the ore by the addition of an alkali or alkaline earth, then leaching the neutralized ore with a cyanide solution, and then passing the cyanide solution containing the precious metal through a sponge of zinc, substantially as set forth.

JOHN STEWART MACARTHUR.
ROBERT WARDROP FORREST.
WILLIAM FORREST.

Witnesses:
ROBERT JAMIESON MACKINLAY,
CHARLES KEITH RITCHIE,
Both of 160 West George Street, Glasgow, Clerks at Law.

# UNITED STATES PATENT OFFICE.

EDWARD D. KENDALL, OF BROOKLYN, NEW YORK.

## COMPOSITION OF MATTER FOR THE EXTRACTION OF GOLD AND SILVER FROM ORES.

Specification forming part of Letters Patent No. 482,577, dated September 13, 1892.

(Application filed May 27, 1892. Serial No. 434,528. No specimens.)

*To all whom it may concern:*

Be it known that I, Edward D. Kendall, a citizen of the United States, residing at Brooklyn, in the county of Kings and State of New York, have invented a new and useful composition of matter to be used for the extraction of gold and silver from ores, so-called "tailings," and other matters containing one or both of these metals, of which composition the following is a specification.

My composition consists of the following ingredients, combined as hereinafter stated: water (hot or cold), potassium ferricyanide (or other soluble ferricyanide), and potassium cyanide (or other soluble cyanide). The best proportions of the two last mentioned constituents vary somewhat with the different ferricyanides and cyanides, and may be determined by calculation based on the molecular weights of the salts or the chemical equivalents of their elements, and also by considering that the purpose of my composition is to set free cyanogen to form—for example, when the potassium salts are used—the soluble double cyanide of gold or silver and potassium, and by applying my theory of the chemical reactions which occur, set forth in the following formula:

$$4\ K_6\ (C_3N_3)_4\ Fe_2 + 16\ KCN + 8\ Au = 4\ K_8\ (C_3N_3)_4\ Fe_2 + 8\ K\ Au\,(CN)_2.$$

The indicated proportions are therefore, practically, by weight, two and one half parts of potassium ferricyanide and one part of potassium cyanide, and these proportions I have found satisfactory in practice; but they may be varied within wide limits without departing from my invention. The potassium ferricyanide, which is a product of the chemical action, facilitates the solution of the resulting cyanides, and after the separation of the precious metal from the menstruum by appropriate means, may be utilized by a process which I propose to make the subject of a separate application.

To prepare my composition, I dissolve the ferrocyanide in one portion of water and the cyanide in another portion, and mix the two solutions; or either salt, in solid form, may by added to the solution of the other. In dissolving the salts I do not always confine myself to a specific proportion of water. More or less water may be used. As a rule, the more concentrated the composition the more energetic its action, but the more costly. Except in treating substances very rich in gold or silver, my composition will always be used in a more or less dilute condition.

In using the herein-described composition, the gold- and silver-bearing minerals, tailings, and other matters, cold or while moderately heated, with or without prior chemical or mechanical treatment, should be placed in tanks, or troughs, or other receptacles made of any suitable material, as wood (if of wood, preferably lined with stoneware slabs), and thoroughly drenched, soaked, or impregnated with my composition, which is, after a time, to be drawn off and washed out, or displaced with water, in order that the contained precious metal may be separated by subsequent operations.

The composition may be used hot or cold. The effect of heat is to hasten the chemical and solvent action.

What I claim, and desire to secure by Letters Patent of the United States, is—

The before-described composition of matter—to be used for extracting gold and silver from minerals, tailings, and other matters containing one or both of these metals—consisting of water, one or more soluble ferricyanides, and one or more soluble cyanides, prepared and combined as herein stated.

EDWARD D. KENDALL.

Witnesses:
EDWARD M. MCCOOK.
S. J. STORRS.

[Second Edition.]

No. 3,024. A. D. 1892.

Date of Application, 16th Feb., 1892.

Complete Specification Left 16th Nov., 1892—Accepted 16th Feb., 1893.

PROVISIONAL SPECIFICATION.

## IMPROVEMENTS IN PRECIPITATING AND COLLECTING METALS FROM SOLUTIONS CONTAINING THEM.

A communication by Bernard Charles Molloy, Member of Parliament, Barrister at Law, of the Middle Temple, London, temporarily residing in Johannesburg, South African Republic.

I, Alfred George Brookes, of 55 and 56 Chancery Lane, in the county of London, Chartered Patent Agent, do hereby declare the nature of this invention to be as follows:

This invention consists in a new method of precipitating and collecting gold and other metals from solutions containing these metals, such as bromide, chloride, and cyanide solutions.

In some cases these solutions are acid, but under the action of this process these solutions are rendered neutral and alkaline, so that the solutions are, or become, alkaline solutions of the metals which are soluble in alkaline solution.

The application of this process may be carried out in apparatus of many forms of construction and under many and differing conditions, depending on the character of the work required to be done.

The following example will, however, explain the nature of the process, and how it may be carried out:

Take a tray or tank constructed of wood, cement, or other suitable material and of a size as may be necessary.

Cover, or partially cover, the bottom of the tank with mercury. On this mercury will rest the solution from which the metals are to be precipitated. This mercury is then charged electrolytically with ammonium, sodium, potassium, or other alkaline metal, which is then amalgamated by the mercury. These alkaline metals, or the amalgams of these metals, when they come to the surface of the mercury and in contact with the water of the solution, decompose the water, the alkaline metal combining with the oxygen of the water to form an alkaline oxide, and so rendering the solution alkaline if not previously so, and the hydrogen of the decomposed water is at the same time evolved in a nascent state from the surface of the mercury in contact with and against the solution from which the metal in solution (say the gold) is to be precipitated and collected.

Under this action the metals in solution (such as gold) will be precipitated and absorbed by the mercury, from which it can be released in the ordinary manner by the action of heat.

In the above-described case the charging of the mercury with the alkaline metal has been effected electrolytically by the electrolysis of say an alkaline salt of the alkaline metal used in a porous vessel in contact with a mercury cathode or other convenient method.

Another though much less advantageous method is the mechanical addition to the mercury, of potassium or other alkaline metal or amalgam of the same, when the nascent hydrogen with an equivalent of the alkaline oxide will be produced.

These are some methods by which the process may be carried out, but there are others, as is evident, which may be employed.

In carrying out this process a solution of bromine, chlorine, or cyanogen may be used to dissolve the gold and other metals from their compounds, and then the process here described may be used to precipitate and collect the metals.

It is evident that generation of nascent hydrogen in contact with an alkaline solution of metals, soluble in such solution, may be obtained by other means, though not detailed here.

Dated the 16th day of February, 1892.

WM. BROOKES & SON,
55 and 56 Chancery Lane, London, Agents for the Applicant.

## COMPLETE SPECIFICATION.

### IMPROVEMENTS IN PRECIPITATING AND COLLECTING METALS FROM SOLUTIONS CONTAINING THEM.

A communication by Bernard Charles Molloy, Member of Parliament, Barrister at Law, of the Middle Temple, London, temporarily residing in Johannesburg, South African Republic.

I, Alfred George Brookes, of 55 and 56 Chancery Lane, in the county of London, Chartered Patent Agent, do hereby declare the nature of this invention and in what manner the same is to be performed, to be particularly described and ascertained in and by the following statement:

This invention consists in a new method of precipitating and collecting the gold and other metals from the solutions containing these metals, such as bromide, chloride, and cyanide solutions, when such are employed for dissolving out the precious metals from compounds containing the same.

In some cases these solutions are acid, but under the action of this process these solutions are rendered neutral and alkaline, so that the solutions are, or become, alkaline solutions of the metals, which are soluble in alkaline solutions.

The application of this process may be carried out in apparatus of many forms of construction, and under many and differing conditions, depending on the character of the work required to be done.

The following example of the treatment of a gold-bearing compound will, however, explain the nature of the process, and how it may be carried out:

The crushed ore is treated, say, with a solution of cyanide of potassium, the quantity and saturation being in proportion to the work to be done. When the solvent solution is sufficiently charged, then the precipitation of the gold, the regeneration of this solvent, and the collection of the gold, is effected as follows:

Take a tray, or tank, constructed of wood, cement, or other suitable material, and of a size as may be necessary. Cover, or partially cover, the bottom of the tank with mercury.

On this mercury will rest, or pass over, the solution from which the metals are to be precipitated. This mercury is then charged electrolytically with ammonium, sodium, potassium, or other alkaline metal. These alkaline metals, when they come to the surface of the mercury and in contact with the water of the solution (now containing gold), decompose the water, the alkaline metal combining with the oxygen of the decomposed water to form an alkaline oxide, rendering the solution alkaline, if not previously so, and the hydrogen of the decomposed water is at the same time evolved in a nascent state from the surface of the mercury in contact with and against the solution from which the metal in solution (say the gold) is to be precipitated and collected. While the current continues to pass, the required nascent hydrogen and oxide will be produced and act on the solution containing the gold.

Under this action the metals in solution such as gold will be precipitated and absorbed by the mercury, from which it can be released in the ordinary manner by the action of heat.

In the above-described case, the charging of the mercury with the alkaline metal has been effected electrolytically by the electrolysis of, say, an alkaline salt of the alkaline metal used in a porous vessel in contact with the mercury cathode or other convenient method.

Another though less advantageous method is the mechanical addition to the mercury of potassium, or other alkaline metal, or amalgam of the same, when the nascent hydrogen, with an equivalent of the alkaline oxide, will be produced.

These are some methods by which the process may be carried out, but there are others, as is evident, which may be employed instead.

The action may be concisely described as follows:

The precious metal is dissolved out say by a solution of potassium cyanide. The solution is then brought into contact with mercury, charged, as described, with potassium. The potassium on coming into contact with the water of the solution decomposes it with the evolution of nascent hydrogen and the formation of the oxide of the alkaline metal. The hydrogen decomposes the solution of the new cyanide of gold—and sets the gold free, which is precipitated upon and collected by the mercury.

The metal of the alkaline oxide reacts upon the cyanogen compound, and so reforms or reproduces the cyanide of potassium. The original solution (of cyanide of potassium) is thus regenerated, and is then ready for re-use, thus effecting a great economy.

The following equations roughly represent the various reactions when cyanogen is the solvent:

$$H_2O + O + 4\,KCN + 2\,Au = 2\,(KAu(CN)_2) + 2\,KOH \quad 2\,(KAu(CN)_2) + 2\,H = 2\,KCN + 2\,CNH + 2\,Au \quad 2\,CNH + 2\,KOH = 2\,KCN + 2\,H_2O.$$

In carrying out this process a suitable solution of a solvent for gold, such as bromine, or chlorine, or cyanogen, or their compounds, may be used to dissolve out the gold and other metals from their ores or compounds, and then the process more particularly described for precipitating the metals in such solution, and in some cases regenerating the solvent solutions and obtaining the metals.

It is evident that generation of nascent hydrogen in contact with an alkaline solution from which metals soluble in such solution are to be precipitated, may be obtained without the employment of a cathode of mercury, and using instead thereof another known cathode.

Although I have hereinbefore indicated some methods by which it may be worked, variations in detail may be effected without departing from the essential features of my invention.

Having now particularly described and ascertained the nature of my said invention, and in what manner the same is to be performed, I declare that what I claim is—

1. The method of obtaining gold and other metals from solutions which have been employed in dissolving out such precious metals from ores, or compounds, containing the same, substantially in the manner and for the purpose hereinbefore set forth.

2. The described method of precipitating and collecting gold and other metals from solutions, such as referred to, containing them, by the action of the alkaline metals, in the manner hereinbefore set forth.

3. The extraction of gold or other metals from ores, or other compounds, by solutions of cyanogen, or its compounds, the precipitation of gold or other metals from such solutions, the regeneration of such solutions, and the collection of the gold or other metal, all substantially as and for the purpose set forth.

4. The described method of precipitating gold and other metals from alkaline solutions, such as indicated, containing them, by the action of nascent hydrogen, in the manner hereinbefore set forth.

5. The precipitation of gold, and other precious metals from solutions, such as indicated, containing them, by means of the alkaline metals, or amalgams of the same, obtained electrolytically, substantially as and for the purpose set forth.

6. The separation and collection of gold or other precious metals from their ores or compounds by means of a suitable solvent for said metals, an electrolyte of the alkaline metals, a current of electricity, a mercury cathode, all substantially as set forth.

7. The employment of bromine, chlorine, iodine, and cyanogen, or their compounds, as solvents for gold or other metals, in combination with the above-described process for effecting the precipitation and collection of the gold and other metals in solution therein, substantially as set forth.

Dated the 16th day of November, 1892.

WM. BROOKES & SON,
55 and 56 Chancery Lane, London, Agents for the Applicant.

[Second Edition.]

No. 12,641. A. D. 1892.

Date of Application, 8th July, 1892.

Complete Specification Left 10th April, 1893—Accepted 8th July, 1893.

## PROVISIONAL SPECIFICATION.

### IMPROVEMENTS IN THE EXTRACTION OF GOLD AND SILVER FROM ORES OR COMPOUNDS CONTAINING THE SAME, AND IN APPARATUS APPLICABLE FOR USE IN THE TREATMENT OF SUCH MATERIALS BY MEANS OF SOLVENTS.

I, John Cuninghame Montgomerie, of the "Water of Ayr" and "Tam O'Shanter" Hone Works, Dalmore, Stair, in the county of Ayr, manufacturer, do hereby declare the nature of this invention to be as follows:

This invention relates to the treatment of auriferous and argentiferous ores or compounds, for the purpose of separating and collecting the gold and silver contained therein, by means of solvent agents—as, for example, cyanide of potassium—and to apparatus applicable for use in processes of this description.

According to the method usually employed in the recovery of gold and silver by means of cyanide of potassium, the ore or other material having been reduced to a finely-triturated state is placed, along with the solvent, in a barrel or other vessel and is there subjected to agitation. After the lapse of a few hours the contents of the barrel are removed to a filter, where the liquid portion of the charge (containing the precious metals in solution) is separated from the ore. The latter is further washed for the removal of any gold or silver remaining (in solution) therewith. The cyanide solution of gold and silver (as also the wash-water) is then treated for the recovery of the precious metals by precipitation.

When a cyanide solvent is employed as hereinbefore described, the proportion of cyanide is necessarily considerably in excess of that required for chemical combination with the gold and silver present in the ore. This excess remains with the liquor after the precious metals have been precipitated therefrom, and is either run to waste or is subjected to a separate process for the recovery of the cyanide.

My improvement in the process of extraction by the method hereinbefore referred to consists in applying the cyanide solution of gold and silver, after having been separated from the ore by filtration, to a subsequent charge or to subsequent charges of fresh ore prior to treating the solution for the separation of the precious metals by precipitation, care being taken that the original quantity of water is made good. With this object the requisite quantity of water is preferably added as soon as the surface of the ore contained in the filter presents a dry appearance, the added water displacing the liquid remaining in the ore and permitting the same (which is highly charged with the solvent and with the precious metals in solution) to be discharged. The solution is then tested for cyanide of potassium (or such other solvent agent as may be employed), and the deficiency supplied by the addition of a suitable quantity of cyanide of potassium (or other solvent agent), thereby restoring the solvent solution to its original strength before adding the same to the fresh charge of ore. Before being added to the fresh charge of ore, the solution is made slightly alkaline by the addition of an alkali, preferably caustic soda.

Where cyanide is employed, it is necessary for the ore to be thoroughly neutralized before treatment; and in some cases it is advantageous to have it slightly alkaline, especially where oxygen is used under pressure. The tailings are then further washed to remove the last trace of gold and silver remaining in solution, and the resultant wash-water is treated in the usual way for the recovery of the precious metals contained therein.

By this mode of procedure considerable economy is effected, both in the quantity of cyanide or other solvent used and in the cost of working, the quantity of liquid subjected to treatment for the recovery of the gold and silver by precipitation being at the same time greatly reduced.

My invention relates secondly to the construction of the barrel or other vessel in which the ore is subjected to the action of the solvent.

If this barrel or vessel be formed of metal, its internal surface is rapidly acted upon by cyanide of potassium or other solvent of the precious metals; and if a lining of wood or similar material be employed, the latter is incapable of withstanding the chemical action of the solvent and the abrasive action of the ore for any length of time.

With a view to overcoming these difficulties, I line the barrel or vessel with tiles or segments composed of glass or glazed porcelain or similar solvent- and acid-resisting material, the same being set in cement adapted to withstand the chemical action of the cyanide or other solvent employed.

My invention relates thirdly to the construction of the filter or leaching vat employed for separating the ore from the cyanide or other solution of gold and silver, or from the wash-water.

A filter constructed according to my improved method comprises an upper vessel for the reception of the mixture of ore and solvent, and a lower vessel in which the solution is received after passing through the filter-bed. The latter is formed of filter cloth carried on wire gauze coated with an acid-proof enamel and supported on wooden laths. The upper vessel is attached to the lower vessel by means of bolts, and is so arranged that the bottom edge of the former rests upon the periphery of the filter cloth and grips the same in a recess formed in the upper edge of the lower vessel, thereby securing a water-tight joint between the two vessels and at the same time holding the filter cloth securely in position.

The filter or leaching vat may be lined with segments or tiles in the manner herein before described with reference to the barrel or other vessel in which the ore is subjected to the action of the solvent.

Dated the 6th day of July, 1892.

G. G. M. HARDINGHAM,
191 Fleet Street, London, E.C., Chartered Patent Agent.

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## COMPLETE SPECIFICATION.

### IMPROVEMENTS IN THE EXTRACTION OF GOLD AND SILVER FROM ORES OR COMPOUNDS CONTAINING THE SAME, AND IN APPARATUS APPLICABLE FOR USE IN THE TREATMENT OF SUCH MATERIALS BY MEANS OF SOLVENTS.

I, John Cuninghame Montgomerie, of the "Water of Ayr" and "Tam O'Shanter" Hone Works, Dalmore, Stair, in the county of Ayr, Scotland, manufacturer, do hereby declare the nature of this invention, and in what manner the same is to be performed, to be particularly described and ascertained in and by the following statement:

This invention relates to the treatment of auriferous and argentiferous ores or compounds, for the purpose of separating and collecting the gold and silver contained therein, by means of solvent agents—as, for example, cyanide of potassium—and to apparatus applicable for use in processes of this description.

According to a method commonly employed in the recovery of gold and silver by means of cyanide of potassium, the ore or other material having been reduced to a finely-triturated state is placed, along with the solvent, in a barrel or other vessel, and is there subjected to agitation. After the lapse of a few hours, the contents of the barrel are removed to a filter, where the liquid portion of the charge (containing the precious metals in solution) is separated from the ore. The latter is further washed for the removal of any gold or silver remaining (in solution) therewith. The cyanide solution of gold and silver, as also the wash-water, is then treated for the recovery of the precious metals by precipitation in a zinc filter or percolator.

When a cyanide solvent is employed as hereinbefore described, a certain portion thereof is taken up by base metals and other impurities invariably present in greater or less proportions in the ore. The solvent is also contaminated by the zinc dissolved whilst the mixture of ore and solvent is under treatment in the zinc percolator; both of these causes resulting in a considerable waste of the cyanide, and in its contamination with deleterious matter.

My improvement in the process of extraction by means of the kind hereinbefore referred to, consists in adding sodium oxide (caustic soda), or other suitable oxide of the alkalies, to the cyanide solution before (or whilst) mixing the same with the ore, thereupon agitating or otherwise treating the resultant mass for the time requisite for enabling the gold and silver to be dissolved by such a solution, then discharging the same into a filter and drawing off the original quantity of water employed, the same being highly charged with the unconsumed cyanide and sodium oxide, and with the precious metals in solution.

By the employment of sodium oxide in the manner hereinbefore described, particularly where the alkali is in excess, I have found that the proportion of cyanide or other solvent may be considerably reduced and an important economy in the cost of working effected.

In carrying out this stage of the process, a sufficient quantity of water is added to the surface of the ore in the filter as soon as it becomes dry, the added water displacing the liquid remaining in the ore and permitting the latter to be discharged. The liquid obtained is then tested for cyanide of potassium and sodium oxide, and the deficiency supplied by the addition of the necessary quantity of these agents so as to restore the solvent solution to its original character and strength. This solution is now applied to a fresh charge of ore, and the same operation is repeated with successive charges till it is found necessary to discharge the solution with a view to precipitating the gold and silver in the usual manner. Experiment alone can determine the quantity of solvent and of sodium oxide appropriate, and the period of time requisite to insure the greatest extraction of the precious metals with the least consumption of the solvent, as these will vary according to the nature of the ore operated upon. (It may be mentioned by

A.D. 1892. JULY 8. N° 12,641.

MONTGOMERIE'S COMPLETE SPECIFICATION.

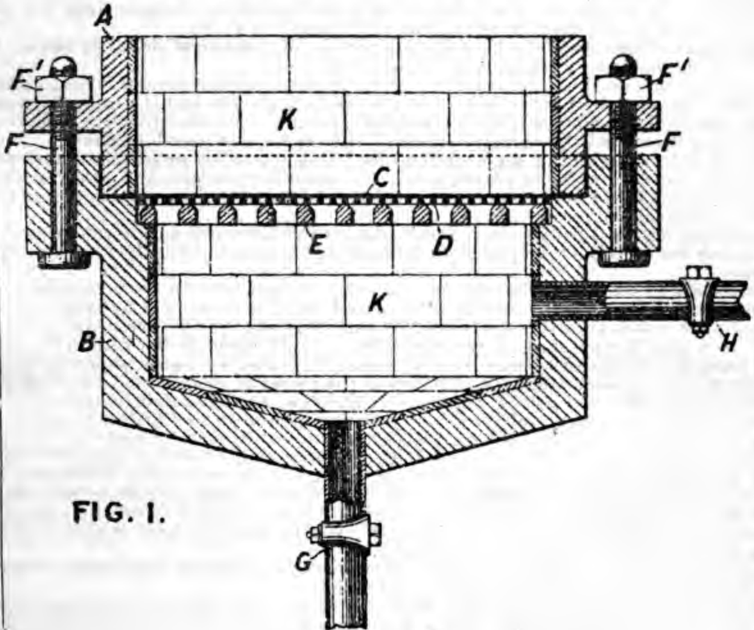



FIG. 1.

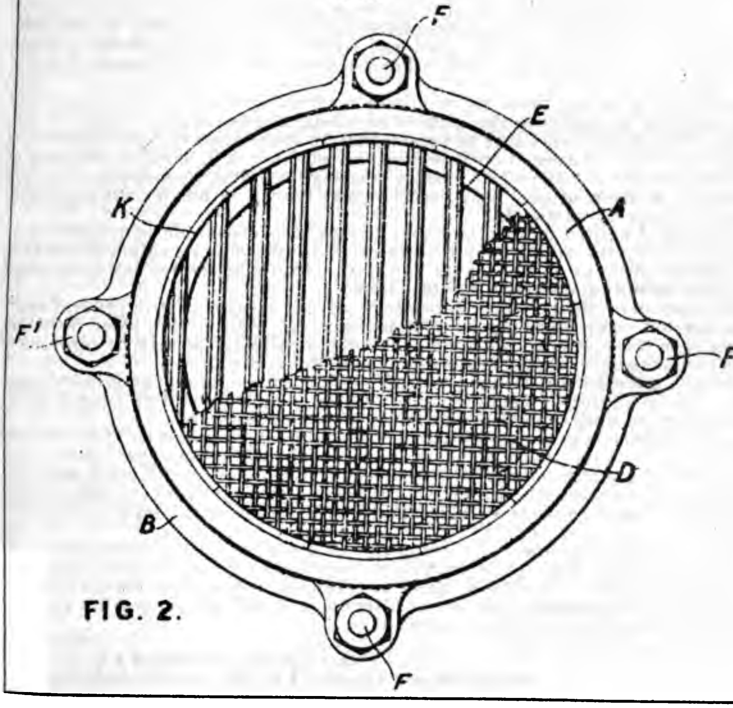



FIG. 2.

[This Drawing is a reproduction of the Original on a reduced scale.]

way of illustration that for an ore containing about 4 ozs. of gold and 12 ozs. of silver per ton, 12 lbs. of cyanide of potassium and 3 lbs. of sodium oxide would be suitable.) The tailings are then further washed to remove the last trace of gold and silver remaining in solution, and the resultant wash-water is treated in the usual way for the recovery of the precious metals contained therein.

By this mode of procedure, the quantity of liquid subjected to treatment for the recovery of the gold and silver by precipitation is greatly reduced.

My invention relates secondly to the construction of the barrel or other vessel in which the ore is subjected to the action of the solvent.

If this barrel or vessel be formed of metal, its internal surface is rapidly acted upon by cyanide of potassium or other solvent of the precious metals; and if a lining of wood or similar material be employed, the latter is incapable of withstanding the chemical action of the solvent and the abrasive action of the ore for any length of time.

With a view to overcoming these difficulties, I line the barrel or vessel with tiles or segments composed of glass or glazed porcelain, or similar solvent- and acid-resisting material, the same being set in cement adapted to withstand the chemical action of the cyanide or other solvent employed.

My invention relates thirdly to the construction of the filter or leaching vat employed for separating the ore from the cyanide or other solution of gold and silver, or from the wash-water.

Apparatus constructed according to my invention is illustrated in the accompanying drawings, whereof Fig. 1 is a vertical section, and Fig. 2 a plan. The apparatus comprises an upper vessel A for the reception of the mixture of ore and solvent, and a lower vessel B in which the solution is received after passing through the filter-bed. The latter is formed of filter-cloth C carried on wire gauze D coated with an acid-proof enamel and supported on wooden laths E on the top of the vessel B. The vessel A is attached to the vessel B by means of bolts F and nuts F[1], and is so arranged that its bottom edge rests upon the circular margin of the filter-cloth C, which it presses against the bottom of a recess or socket formed in the upper edge of the vessel B, thereby securing a water-tight joint between the two vessels and at the same time holding the filter-cloth securely in position. G is a draw-off cock; H being the exhaust cock. The vessels A and B are lined with segments or tiles K, composed of glass, glazed porcelain, or similar solvent- and acid-resisting material, set in cement adapted to withstand the chemical action of the cyanide or other solvent employed. When a barrel is used, it may be lined with segments or tiles set in a similar manner.

Having now particularly described and ascertained the nature of this invention, and in what manner the same is to be performed, I claim—

1. The improved process of extracting gold and silver from ores or compounds containing the same, substantially as herein described; the same consisting in mixing the ore with a solution of cyanide of potassium or other cyanide solvent rendered alkaline by the addition of sodium oxide or an equivalent alkaline oxide, filtering or otherwise separating the liquid (containing the gold and silver in solution) from the ore, and treating the former, by precipitation or other known mode, for the recovery of the precious metals.

2. In the extraction of the precious metals by a solvent process of the general character herein referred to, applying the solvent solution, after separation from the first charge of ore, to a subsequent charge or successively to subsequent charges of fresh ore, the solution being fortified at each operation by the addition of a suitable quantity of the chemical agents employed, and ultimately treating the liquid (consisting of a more or less saturated solution of gold and silver) by any known means for the separation and recovery of the precious metals.

3. In the process of extracting gold and silver by means of cyanide of potassium or other cyanide solvent, the addition of sodium oxide or other suitable alkaline oxide to the solvent, either prior to or during its admixture with the ore, for the purpose of economizing the solvent and expediting its action.

4. In the extraction of the precious metals by a solvent process of the general character herein referred to, discharging the solvent remaining with the ore after filtration by adding water to the surface of the ore and thereby displacing the solvent containing the precious metals in solution, substantially as herein described.

5. In apparatus adapted for use in the treatment of ores or compounds containing gold or silver, a barrel, filter, or leaching vessel such as A or B lined with tiles K set in an acid- or solvent-resisting cement, substantially as herein described.

6. The herein described apparatus for use in the treatment of ores or compounds containing gold and silver by means of solvents, the same comprising an upper vessel A for the reception of the ore and solvent, a lower vessel B in which the solution is received, a filter cloth C held between the lower part of the vessel A and a socket in the upper part of the vessel B, wire gauze D on which the filter cloth lies, and bars E for supporting the wire gauze.

7. The herein described apparatus for use in the treatment of ores or compounds containing gold and silver by means of solvents, the same comprising an upper vessel A lined with tiles K, a lower vessel B also lined with tiles K, a filter cloth C held between the vessels A and B, wire gauze D under the filter cloth, bars E for supporting the wire gauze, a draw-off cock G, and an exhaust cock H.

Dated the 8th day of April, 1893.

G. G. M. HARDINGHAM,
191 Fleet Street, London, E.C., Chartered Patent Agent.

# UNITED STATES PATENT OFFICE.

ALEXIS JANIN AND CHARLES W. MERRILL, OF SAN FRANCISCO, CALIFORNIA.

## PROCESS OF LEACHING ORES WITH SOLUTIONS OF ALKALINE CYANIDES.

Specification forming part of Letters Patent No. 515,148, dated February 20, 1894.

(Application filed June 12, 1893. Series No. 477,338. No specimens.)

*To all whom it may concern:*

Be it known that we, Alexis Janin and Charles W. Merrill, citizens of the United States residing in the city and county of San Francisco, State of California, have invented an Improvement in Processes of Leaching Ores with Solutions of Alkaline Cyanides; and we hereby declare the following to be a full, clear, and exact description of the same:

Our invention relates to an improvement in the art of leaching ores with solutions of alkaline cyanides, and consists in, first, precipitating and separating, in the form of silver sulphide, by means of an alkaline sulphide or of sulphuretted hydrogen gas, all or the greater portion of the silver dissolved from the ore by such solutions, and then precipitating in the metallic state, by means of metallic zinc, the gold contained in the same solution, together with any silver which has escaped precipitation as a sulphide.

In the usual method of leaching ores with a solution of potassium cyanide, the gold and silver extracted are both precipitated from the solution in the metallic state, with metallic zinc. When much silver is present, this method involves a large consumption of zinc and consequent contamination of the cyanide solution by the zinc dissolved, and unless the contact between the zinc and the silver-bearing solution be greatly prolonged, the precipitation of the silver is imperfect. Furthermore, the potassium cyanide which combines with the zinc dissolved is practically lost. When the silver is precipitated from its solution in potassium cyanide by means of an alkaline sulphide an alkaline cyanide is regenerated, which is again available for leaching. If sulphuretted hydrogen gas be used to precipitate the silver there is also formed free hydrocyanic acid, but if the solution of potassium cyanide contains free alkali, or if such be added to the solution, no free hydrocyanic acid will escape, either because the sulphuretted hydrogen gas first combines with the alkali, to form a sulphide which precipitates the silver in the manner described, or because any hydrocyanic acid generated will also combine with the free alkali to form an alkaline cyanide.

We have found that, whereas, silver is not precipitated at all, or only very imperfectly, from strong solutions of potassium cyanide, by means of the agents hereinafter mentioned, yet when the silver-bearing solution contains only about one and one half per cent, or less, of pure potassium cyanide (KCN) or its equivalent, then the silver can be thoroughly precipitated by means of the sulphides of sodium, potassium, or ammonium, or by sulphuretted hydrogen gas, and the precipitation of the silver becomes more imperfect as the strength of the solution in potassium cyanide is increased. Therefore when leaching silver-bearing ores we employ solutions containing, at the most, two per cent of potassium cyanide or its equivalent. As a precipitating agent we employ preferably a solution of sodium sulphide, approaching, as nearly as practicable, to the composition of a monosulphide, in order to avoid, as much as possible, the separation of free sulphur in precipitating the silver.

In practice we leach ores containing both gold and silver, with a solution of potassium cyanide containing not more than two per cent of KCN or its equivalent, or as much weaker as is consistent with a thorough extraction. The solution, after passing through the ore, is run into precipitating vats, where a solution of sodium sulphide is added in sufficient quantity to convert the silver present into sulphide of silver, or in a little less than that amount, in order to avoid the possibility of any excess of the precipitating agent remaining in the solution which might be prejudicial in its further use. The precipitate of silver sulphide is allowed to settle, the supernatant solution of potassium cyanide is then drawn off, and the gold, together with any silver remaining in solution, is precipitated by means of metallic zinc.

Having thus described our invention, what we claim as new, and desire to secure by Letters Patent, is—

The improvement in the art of leaching ores with solutions of alkaline cyanides, which consists in first leaching the ore with such solutions, then adding to the solution an agent which will precipitate the silver present as a sulphide, and then precipitating the gold in the solution with metallic zinc, substantially as herein described.

In witness whereof we have hereunto set our hands.

ALEXIS JANIN.
CHARLES W. MERRILL.

Witnesses:
S. H. NOURSE.
WM. F. BOOTH.

# NEW ZEALAND PATENT OFFICE.

JOHN STEWART MACARTHUR AND CHARLES JAMES ELLIS.

**APPLICATION FOR LETTERS PATENT FOR IMPROVEMENTS IN EXTRACTING GOLD AND SILVER FROM ORES AND THE LIKE.**

We, John Stewart MacArthur, Managing Director of the Cassel Gold Extracting Company (Limited), and Charles James Ellis, technical chemist to the said company, both of 157 West George Street, Glasgow, in the county of Lanark, North Britain, do declare the nature of our invention for "Improvements in extracting gold and silver from ores and the like," and in what manner the same is to be performed, to be particularly described and ascertained in and by the following statement:

Our said invention relates to what is known as the "MacArthur-Forrest process" for extracting gold and silver from ores and the like by means of cyanides, and has for its object to increase the efficiency and economy of that process in cases in which, from the nature of the ores treated, or other circumstances, it is found that in the solution of cyanide as heretofore used there is formed, or becomes present, a sulphide soluble therein which retards and objectionably affects the action of the cyanide on the precious metals.

Our invention consists in removing or rendering inert such sulphide by adding to the solution of cyanide, or to the ore, or to the mixture of ore and cyanide solution, a suitable salt or compound of a metal which will form with the sulphur of the sulphide a sulphide which is practically insoluble or inert in the cyanide solution, or which will materially diminish the objectionable action.

In carrying out our invention, we may use any one or more of various metallic salts or compounds, of which the following may be mentioned, by way of example, preference being given to them in the order in which they are noted, namely: Salts or compounds of lead, such as plumbates, carbonate acetate, or sulphate of lead; or salts or compounds of other metals, such as sulphate or chloride of manganese, zincates, oxide, or chloride of mercury, and ferric hydrate, or oxide. The proportion to be used in any case will depend on the proportion of soluble sulphide which has to be dealt with in the cyanide solution applied to the particular ore, and is easily and most conveniently ascertained by trials of a few samples in each case. In the case of some ores containing sulphur, we find the addition of salts or compounds, as and for the purpose hereinbefore referred to, and especially those of lead and mercury, increases the percentage of precious metals obtained.

Having now particularly described our said invention, and in what manner the same is to be performed, we declare what we claim is—

1. In the MacArthur-Forrest process for extracting gold and silver from ores and the like, the addition to the cyanide solution, or to the ore, or to the mixture of ore and cyanide, of salts or compounds of lead, substantially as and for the purposes hereinbefore described.

2. In the MacArthur-Forrest process for extracting gold and silver from ores and the like, the addition to the cyanide solution, or to the ore, or to the mixture of ore and cyanide, of any one or more of the metallic salts or compounds hereinbefore indicated, and capable of forming insoluble sulphides, as and for the purposes hereinbefore described.

Dated June 29, 1893.

JOHN STEWART MACARTHUR.
CHARLES JAMES ELLIS.

# NEW ZEALAND PATENT OFFICE.

## CARL MOLDENHAUER.

### APPLICATION FOR LETTERS PATENT FOR IMPROVEMENTS IN RECOVERING GOLD AND OTHER PRECIOUS METALS FROM THEIR ORES.

I, Carl Moldenhauer, of Frankfort-on-Main, in the Empire of Germany, do hereby declare that the nature of my invention for improvements in recovering gold and other precious metals from their ores, and the manner in which the same is to be used, are particularly described and ascertained in and by the following statement:

This invention relates to extracting gold and other precious metals from their ores by means of a solution of cyanide of an alkali or an alkaline earth, and has for its object to render the process more expeditious and considerably cheaper than heretofore.

The invention consists, firstly, in adding to the cyanide solution an artificial oxidizing agent; and secondly, in precipitating the extracted precious metal out of its cyanide solution by means of aluminium or alloys or amalgam thereof.

As to the first part of my invention, I have found that the dissolving action of the cyanide solution on the precious metal is highly expedited, and much cyanide is saved, if an artificial oxidizing agent is added to the said cyanide solution. As such an artificial oxidizing agent, I use, by preference, ferricyanide of potassium, or another ferricyanogen salt of an alkali or of an alkaline earth. In either case the ferricyanogen salt is preferably employed in alkaline solution. The result of this addition of an artificial oxidizing agent is that the dissolving action of the solvent is rendered more energetic, and consequently a considerably smaller quantity of the solvent is required. Thus, by the addition of ferricyanide of potassium or other ferricyanogen salt in alkaline solution, as much as 80 per cent of the potassium may be saved. The proportions preferred are from one half to two parts of ferricyanide to one part of cyanide.

It may be remarked that the ferricyanide of potassium alone will not dissolve the gold, and does not, therefore, come under the category of the solvent heretofore employed in processes of extraction. It does not, therefore, render unnecessary the employment of the simple cyanide as a solvent, but only reduces the amount required, owing to the capacity of the ferricyanide to act as an oxidizing agent; consequently the cyanogen of the ferricyanide is not used to form the gold cyanide compound. I may also employ permanganate of potash as the oxidizing agent instead of the ferricyanide. The said permanganate of potash is also added in solution and in the same proportions as before, namely, from one half to two parts of permanganate to one part of cyanide.

In lieu of permanganate of potash, any other suitable oxidizing agent can be used in carrying out my invention in practice, the invention not being restricted to the use of any special oxidizing agent, but includes the use of an agent that exerts an oxidizing action in the cyanide solution. The process can be carried out in a ball-mill lined with porcelain, wood, or other substance unattackable by the chemicals employed.

I may also use the cyanide solution and the oxidizing agent in combination with a preliminary treatment of the ore with any acid or salt that renders the precious metal better adapted to the subsequent treatment of the cyanide solution.

The second part of the process consists in precipitating the dissolved gold or precious metal out of its cyanide solution by means of aluminium alloy or aluminium amalgam; but this can also be applied with advantage to sulphurized solutions containing free alkali—that is to say, solutions which contain gold in the form of sulphuret, or hyposulphide of gold.

Zinc has heretofore been employed in practice by preference in precipitating gold from the cyanide solutions obtained by leaching auriferous ores. The employment of zinc for this purpose is found, however, to be attended with serious disadvantages. Now, I have discovered that aluminium can be employed for this purpose in place of zinc, without the disadvantages attending the use of the latter.

Whilst zinc forms a combination with the bound or free compound of cyanogen and alkali contained in the cyanide solution, aluminium separates the gold very quickly from the cyanogen solution without entering into combination with the cyanogen, but simply reacting with the caustic alkali which is present at the same time. By the action of aluminium the cyanide of potassium employed for leaching the gold out of its ore is regenerated, which is not the case when zinc is employed. But the zinc does not confine itself to entering into combination with the cyanogen of the cyanogen compounds of the gold, but also acts upon the free cyanide of potassium contained in the solution, so that a great part of the latter is consumed; but this is not the case when aluminium is employed.

These results are of the greatest importance when the solutions separated from the gold is to be employed in subsequent gold-extracting operations, as the whole of the cyanogen in the regenerated and liberated cyanide of potassium is enabled to renew its action; but the lyes resulting from the employment of zinc cannot be employed with

the same advantage in subsequent operations for the extraction of gold. Numerous attempts have been made to regenerate the zinc, but are found to be inconvenient and costly. It is consequently evident that an important saving in cyanide of potassium is obtained by the employment of aluminium.

Aluminium acts in a like manner in a sulphurized alkaline solution—that is to say, in a solution containing the gold in the form of sulphuret of gold or hyposulphide of gold. It does not enter into combination with the sulphur in a solution of this description. This great and important advantage attending to the employment of aluminium, aluminium alloys, or aluminium amalgam, is combined with other advantages, as follows:

Aluminium is far less subject to oxidation than is zinc, so that it can be sent from its place of production in the form in which it is to be used for the precipitation, whereas when zinc is employed it is considered to be an important advantage to reduce it to the required form at the place where it is employed, and immediately before using it. For the same reason, the repeated employment of the aluminium is admissible for continuous precipitation.

Finally, the quantity of aluminium required for precipitating the same quantity of precious metal is about four times less than the amount of zinc required to produce the same effect.

I am aware that attempts have been made to employ aluminium for precipitating precious metals from acid or neutral solutions, but in this case it offers no advantages as compared with zinc and iron.

On the other hand, the practical precipitation of precious metals from alkaline cyanide solutions or sulphurized solutions by means of aluminium was not known, neither was it known that by the employment of the same in the presence of free alkali it was possible to obtain the important advantages hereinbefore set forth.

Of course, instead of pure aluminium, alloys or an amalgam thereof can be used with a like advantage; furthermore, I do not confine myself to the use of the aluminium, its alloys or amalgam, in any special form, as it may be used in any suitable form without departing from my invention.

Having particularly described and ascertained the nature of my said invention, and in what manner the same is to be performed, I declare that what I claim is—

1. Extracting gold and other precious metals from their ores by subjecting the ores to the dissolving action of a cyanide of an alkali or an alkaline earth in the presence of an oxidizing agent, substantially as and for the purpose hereinbefore set forth.

2. Extracting gold from its ores by subjecting the ores first to the action of an acid and subsequently to the dissolving action of cyanide of an alkali or an alkaline earth in the presence of an oxidizing agent, substantially as and for the purpose hereinbefore set forth.

3. Extracting gold from its ores by subjecting the ores to the dissolving action of cyanide of an alkali or an alkaline earth in a ball-mill, substantially as and for the purpose hereinbefore set forth.

4. Precipitating gold or other precious metals out of their solutions by means of aluminium, aluminium alloys, or aluminium amalgam, in the presence of a free alkali, substantially as hereinbefore described.

Dated August 31, 1893.

CARL MOLDENHAUER.

# NEW ZEALAND PATENT OFFICE.

## CARL MARIA PIELSTICKER.

### APPLICATION FOR LETTERS PATENT FOR IMPROVEMENTS IN THE EXTRACTION OF GOLD AND SILVER FROM ORES.

I, Carl Maria Pielsticker, of No. 43 Connaught Road, Harlesden, in the county of Middlesex, England, engineer, do hereby declare the nature of my invention for Improvements in the Extraction of Gold and Silver from Ores, and in what manner the same is to be performed, to be particularly described and ascertained in and by the following statement:

My invention has for its object the extraction of gold and silver, particularly from sulphide, and from such ores in which the precious metals exist in a state of extremely fine division, and it consists essentially in treating the powdered ore with a solution of cyanide of potassium or a cyanide or cyanogen-yielding substance in conjunction with an electric current, continuous circulation of the solvent, continuous precipitation of the dissolved precious metals by electrolysis, and continuous regeneration of the solvent.

In carrying out my invention, I employ a tank, marked A on the accompanying drawing, which I call the ore-tank, in which the ore is subjected to the treatment with cyanide of potassium in conjunction with an electric current. About 3 inches from the bottom I place a perforated plate, H, preferably of iron or carbon, covered with a porous material, such as cocoanut matting. The plate H serves as anode, and is insoluble, or practically so, in cyanide of potassium.

If made of iron, I prefer a highly carburetted iron, or ore containing a high percentage of silicum. Near the top of the ore-tank I place a second perforated plate, G, which serves as cathode. Both these plates are connected by means of insulated wires, e—e, with a dynamo, D, or other source of electricity.

The ore-tank A is connected near its top by means of a pipe with a second tank, B, containing a number of baffle plates, K, or their equivalent, which are destined to arrest any suspended matter flowing over with the solution from the ore-tank, and which otherwise would greatly interfere with the precipitation of the dissolved precious metals in the following tank, C, connected with the tank B near the top by means of a pipe. The precipitating-tank C contains one or more pairs of electrodes, M and N, connected with the dynamo D, or other source of electricity, by means of the insulated wires, g and $g^1$, of which the anode preferably consists of carbon or other material, insoluble, or practically so, in cyanide of potassium. A pump P is connected with the ore-tank A under the anode H on the one hand, and with the top of the depositing-tank C on the other hand, enabling me to maintain a circulation of the solvent through the set of tanks.

In operating my invention, I fill the ore-tank A between the electrodes H and G with powdered ore, and admit into the three tanks A, B, and C, a solution of cyanide of potassium, filling them above the level of the pipes which connect one tank with the other. The strength of the cyanide solution may vary, care being taken to have sufficient cyanogen present to bring the gold and silver in the ore into the solution, the amount of which has previously been ascertained by assay. I connect the electrodes H and G in the ore-tank A, and M and N in the depositing-tank C, with the dynamo D, or other source of electricity, and force the cyanide solution from below through the ore in the tank A.

The solution pregnant with dissolved precious metals overflows into the settling-tank B, where it clears itself from suspended matter, and becomes thus fit to part with the dissolved precious metals on overflowing into the depositing-tank C, where the latter are precipitated on the cathode, and from which they are recovered by amalgamation or otherwise. The cyanide solution, freed from dissolved metals, and therefore in a better condition to dissolve more metal than when loaded with metal in solution, is pumped from the depositing-tank C, again through the ore in the tank A, where it dissolves a fresh proportion of precious metals, and so on, continuously, until the precious metals contained in the ore under treatment are dissolved.

In this manner my process becomes a continuous one, of dissolving the precious metals from the ore, preparing the solution pregnant with dissolved metals for electrolysis by separating continuously the suspended matter therefrom, precipitating continuously the dissolved metals by electrolysis, and regenerating continuously the solvent for further action on the undissolved precious metals still contained in the ore. Very little of the precious metals are precipitated on the cathode G in the ore tank, as the amount of the suspended matter present in the solution interferes with precipitation in this tank.

The electric current in the depositing-tank must be of low tension, and so regulated as to be of just sufficient strength to deposit the gold and silver without also decomposing the cyanide of potassium; the gold and silver, being more readily precipitated from their double salts of cyanide of gold (or silver) and potassium than the cyanogen, is set free from the simple salt of cyanide of potassium so long as the current of electricity is sufficiently low in tension, and so long as there are metals present in the solution.

The original solution can therefore be used over and over again for a long time, and only the loss made good occasionally.

In practice I find that an electro-motive force of about one volt, and an intensity of about ten ampères per square meter of surface of cathode, is well adapted for depositing the gold and silver in the tank C. I may find it desirable to employ a current of electricity of greater potential in the ore-tank A and of lesser potential in the depositing-tank C.

The great advantage in treating ores with cyanide of potassium in conjunction with an electric current lies in the fact that the precious metals are attacked by the cyanide solution more energetically in conjunction with a current of electricity than without one; further, when the dissolved precious metals are precipitated by means of an electrical current and an insoluble anode very little cyanide and no metal is consumed, as is the case when, for instance, zinc is used as a precipitant, when not only zinc is consumed, but also cyanide of potassium in the formation of a double salt of cyanide of zinc and potassium. Moreover, serious losses in gold and silver are occasioned in the recovery of the precious metals from the zinc slimes, whereas nothing can be simpler than their recovery from the cathode by amalgamation. Again, the precipitation of gold and silver is greatly accelerated by the electric current.

When these metals are precipitated by zinc without a current of electricity, the latter goes into solution as a double salt of cyanide of zinc and potassium, but the amount of zinc which is converted into cyanide of zinc is directly proportionate to the time during which it is in contact with the cyanide solution. Therefore, the more time is consumed in precipitating the gold and silver, the more cyanide and the more zinc will be wasted.

The cyanide process is most advantageously employed on ores in which either the gold is so finely divided in a free state that it is difficult to retain it by older methods, or for sulphide ores. Free gold is certainly more quickly dissolved by cyanide of potassium in conjunction with an electrical current than without one. As regards pyritic ores, if they are simply iron pyrites, as they are in a great number of cases, a cyanide of potassium solution, whatever its strength may be, has as little action on them when used in conjunction with an electric current of the strength I use as without one, only the gold and silver in the ore are more quickly dissolved in conjunction with an electrical current than without one.

If the ores contain sulphides, oxides, or carbonates, for instance, of copper and zinc, these are as easily dissolved by a cyanide of potassium solution, whether employed by itself or in conjunction with an electric current such as I use. Such ores, however, I prefer to treat first with, say, a 5 per cent sulphuric acid or other acid solution in water, or a strong solution of sulphurous acid in water in sufficient quantity to dissolve such metals, then leach with water, and then treat with the cyanide solution in conjunction with the electric current.

I would have it understood that I do not limit myself to the precise details herein set forth and illustrated on the drawing—for example, the number, nature, and position of electrodes, of the sources of electricity, and of the number, shape, and position of the tanks; all may be varied while retaining the construction and combinations for the proper carrying out of my process of extraction of gold and silver from their ores; further, I am aware that cyanide of potassium has been used in conjunction with an electric current for like purposes, and I make no broad claim thereto.

Having now particularly described and ascertained the nature of my said invention, and the manner in which the same is to be performed, I declare that what I claim is—

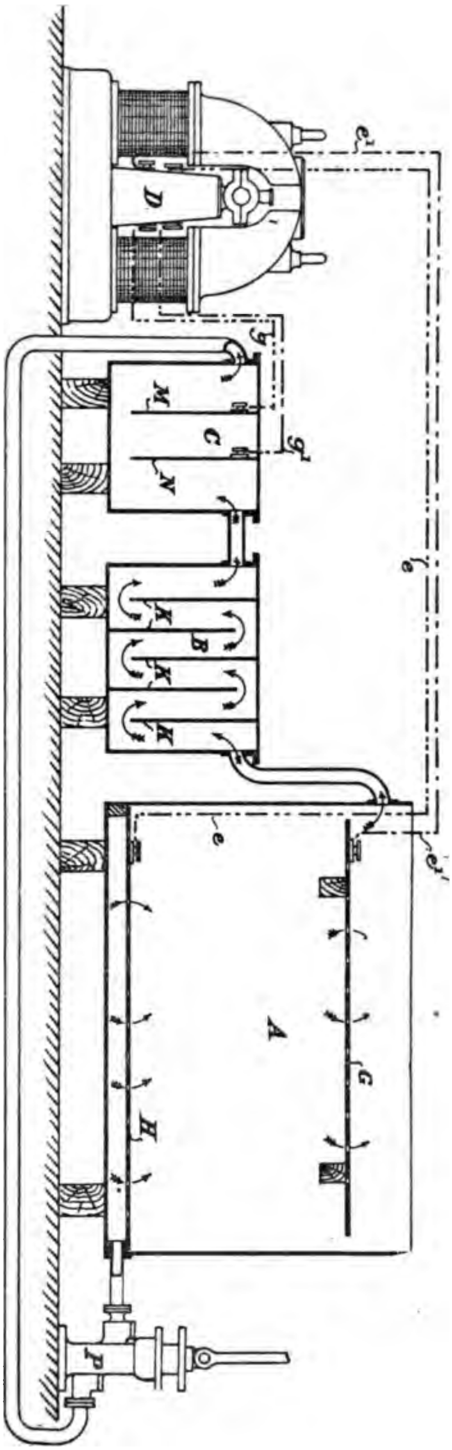
1. The process of separating gold and silver from their ores, which consists in treating the powdered ore with a solution of cyanide of potassium in conjunction with an electric current, depositing the precious metals constantly by means of a current of electricity of low tension and electrodes, of which the positive one is insoluble in cyanide of potassium, and bringing the cyanide of potassium solution thus freed from dissolved metal constantly again into contact with the ore, whereby I obtain a continuous process of extraction and precipitation, all substantially as herein described.

2. In the process of separating gold and silver from their ores by means of a solution of cyanide of potassium in conjunction with an electric current, bringing the cyanide of potassium solution freed from dissolved metals continuously into contact with the ore, substantially as described.

3. In the above-described process of separating gold and silver from ores by means of a solution of cyanide of potassium in conjunction with an electric current, depositing the dissolved metals by means of electrodes contained in depositing-tank or tanks, an electric current being passed through the ore-tank and depositing-tank, substantially as set forth.

4. In the above-described process of separating gold and silver from their ores by means of a solution of cyanide of potassium in conjunction with an electric current, treating the ore with an acid in combination with a subsequent treatment of cyanide of potassium in conjunction with an electric current and continuous circulation of the solution, substantially as described.

5. In the above-described process of separating gold and silver from their ores by means of a solution of cyanide of potassium in conjunction with an electric current, subjecting the ore and solution in the ore-tank to an electric current of greater potential, and depositing the dissolved metal in a depositing-tank by an electric current of lesser potential, substantially as described.




IMPROVED APPARATUS FOR THE EXTRACTION OF GOLD & SILVER FROM ORES.

— C. M. Pielsticker's Patent. —

6. In the above-described process of separating gold and silver from their ores in conjunction with an electric current, the use of a current of electricity of sufficient strength to decompose the double salt of cyanide of gold or silver and potassium without decomposing the cyanide of potassium itself.

7. In the above-described process of separating the gold and silver from their ores in conjunction with an electric current, the combination of an ore-tank with a settling-tank and a depositing-tank, substantially as described.

CARL PIELSTICKER.
By W. H. QUICK, His Agent.

Dated this 14th day of December, 1893.

# UNITED STATES PATENT OFFICE.

WILLIAM DAVID JOHNSTON, OF SAN FRANCISCO, CALIFORNIA.

**[ME]THOD OF ABSTRACTING GOLD AND SILVER FROM THEIR SOLUTIONS IN POTASSIUM CYANIDES.**

Specification forming part of Letters Patent No. 522,260, dated July 3, 1894.

(Application filed November 20, 1893. Serial No. 491,473. No specimens.)

*[To a]ll whom it may concern:*

[B]e it known that I, William David Johnston, a citizen of the United States, residing [in t]he City and County of San Francisco, State of California, have invented an Improve[men]t in Methods of Abstracting Gold and Silver from their Solutions in Potassium [Cya]nide; and I hereby declare the following to be a full, clear, and exact description of [the] same:

[H]eretofore when solutions of gold and silver have been made in potassium cyanide, [the] metals have been recovered from their solution by the use of zinc in various forms. [T]he object of my invention is to recover the metals in a shorter time, and more eco[no]mically, by the use of pulverized carbon, preferably in the form of charcoal.

[T]o carry my invention into effect, I take carbon in a pulverized form as above, and [pla]ce it upon suitable supports so as to form it into filters, through a series of which the [cya]nide liquid is caused to pass successively, leaving the metal deposited upon the [car]bon. The gold and silver are then recovered from the carbon by carefully burning [the] carbon, and smelting the residue with the usual fluxes. By thus employing a series [of f]ilters through which the solution is passed successively, I am able to recover upward [of 9]5 per cent of the precious metal contained in the solution.

[W]hen only one filter is employed, only about one fourth of the gold can be extracted.

[H]aving thus described my invention, what I claim as new, and desire to secure by [Let]ters Patent, is—

[1.] The process of abstracting gold and silver from their solution in potassium cyanide, [con]sisting in passing the liquid through a series of carbon filters within which the gold [is a]rrested, substantially as described.

[2.] The process of abstracting gold and silver from their solution in potassium cyanide, [con]sisting in passing the liquid through a series of carbon filters within which the gold [is a]rrested, and then recovering the metal by burning the carbon and smelting the [res]idue with suitable fluxes, substantially as described.

[I]n witness whereof I have hereunto set my hand.

WILLIAM DAVID JOHNSTON.

[W]itnesses:
S. H. NOURSE.
H. F. ASCHECK.

NOTE.—The patents, where the country is not mentioned, are to be [un]derstood as being issued in Great Britain.

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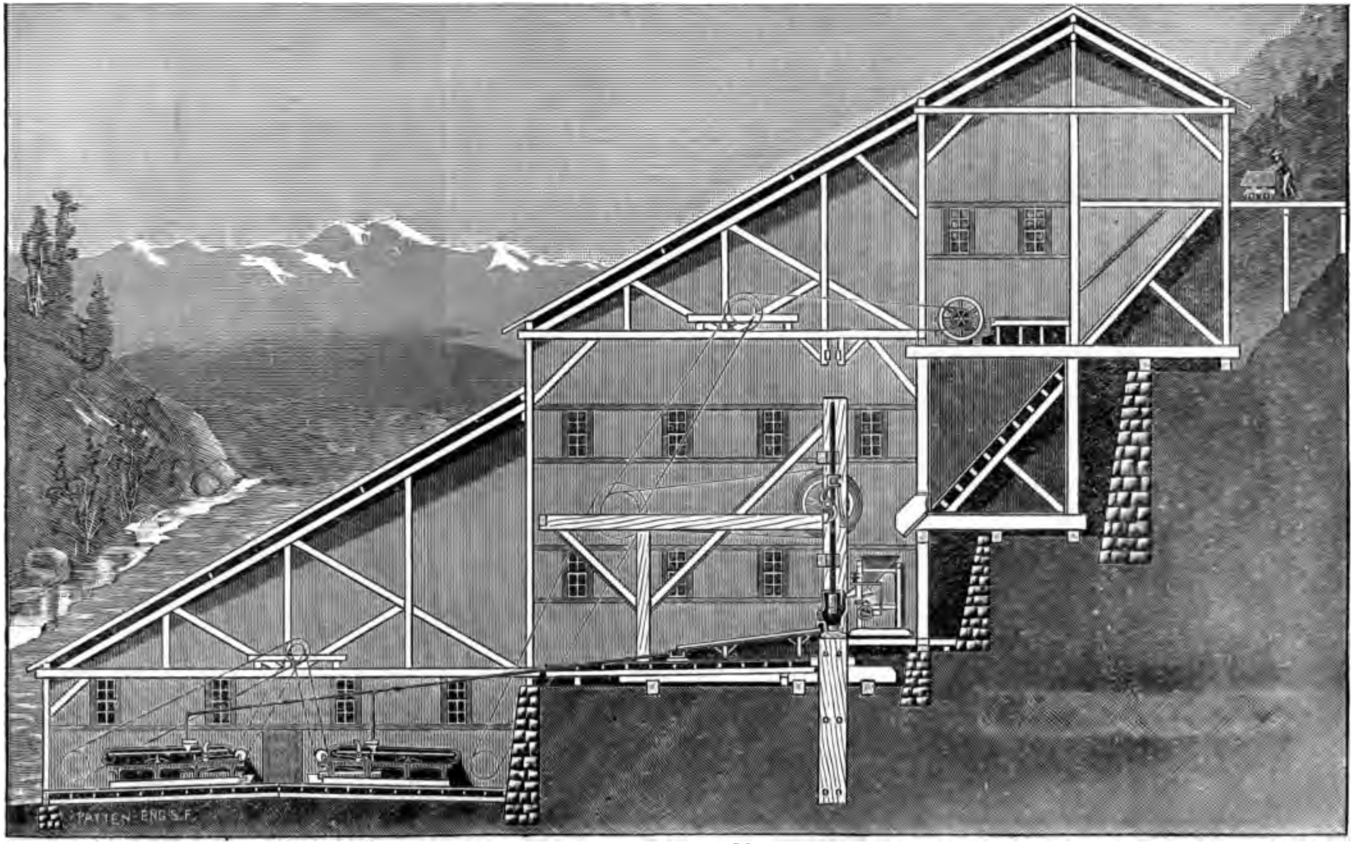


A Modern Gold Mill.

CALIFORNIA STATE MINING BUREAU.

J. J. CRAWFORD, State Mineralogist.

BULLETIN No. 6. San Francisco, September, 1895.

# CALIFORNIA GOLD MILL PRACTICES.

By ED. B. PRESTON, M.E.,

FIELD ASSISTANT.

SACRAMENTO:
A. J. JOHNSTON, : : : : : : SUPERINTENDENT STATE PRINTING.
1895.

*Hon.* J. J. CRAWFORD, *State Mineralogist:*

In preparing, at your request, this article on the California Gold Mill Practices, I have endeavored to furnish something that should be of practical benefit to actual working millmen who might not come in reach of expensive works on the subject, rather than a further contribution to the literature on gold milling. A demand for this kind of a work is evidenced by the constant inquiry at the State Mining Bureau for former Reports dealing in part on this subject.

To W. H. Storms, M.E., my acknowledgments for valuable data collected in the counties of Amador, Calaveras, Tuolumne, and Mariposa are due.

Some of the illustrations are repeated from former Bureau publications, and for many of the others I am indebted to the principal San Francisco iron works, including the Union, Risdon, Hendy, Fulton, and the Pelton Company.

ED. B. PRESTON, M.E.,
Field Assistant.

SAN FRANCISCO, CAL., January, 1895.

# NOTES ON GOLD MILLING IN CALIFORNIA.

The system of reduction of gold-bearing ores with stamps, as at present carried out in California, is the result of progressive improvement during the past forty-four years. The first successful mill in this State was built in the winter of 1850–51, and used steam for power.

Starting with the ancient Mexican arrastra, crushing, with the help of a mule and one man, a few hundred pounds of ore at a charge, we have progressed to the present aggregation of mechanical appliances, as seen in the modern stamp-mill, requiring great motive power, and disposing of hundreds of tons of ore in the course of a day. This progression is largely the result of accumulated practical experience on the part of the designers and builders of mills, as well as of the millmen in the handling of the various gold ores. Of late years scientific investigations have greatly aided in improving both the process and the mechanism.

That the results accomplished have been of economic value is evident from the fact that while formerly a yield of 30% to 40% of the total gold in the ore was the average obtained, the best mills of to-day are able to more than double these figures. That our methods may still be improved upon, and the margin of wasted gold be further narrowed down, is the point for which all intelligent millmen are striving. While the stamp-mill itself had been used for crushing ores long before the discovery of gold in California, since that time it has been greatly improved in detail, and its capacity and efficiency increased, hence what is now known as the "California gold mill" is a very different affair from the clumsy mills first used for crushing quartz in this State. The California gold-milling processes and the California millmen are, as a result, finding due recognition outside of their own immediate field of operation, as is evidenced by the increasing outside and foreign demand for our men and milling machinery. The development of the milling process, keeping pace with the improvement of the machinery required for ore reduction, has had the beneficial effect of greatly lessening the working expenses, permitting ores of a low grade to be worked at a profit. California has a great abundance of this class of ores, comparatively untouched, and these must be mainly relied on in the future as the sources of the precious metal. Already, under extremely favorable conditions, ores are being mined and milled in California at a cost of 50 cents per ton, as at the Spanish Mine, in Nevada County, where, with Huntington roller mills, ores yielding 85 cents per ton have been worked at a profit.

### MILL SITE.

When assured of a constant and sufficient supply of ore, it is of the greatest importance that the site for the mill should be chosen with due regard for economic treatment. This necessitates the observance of the following points: The means of transportation of the ore from the mine

to the mill, which should be done automatically, or at least with as little handling as possible, conveying the ore at once to the highest point in the mill, so that it will descend by gravity from one to the other in all the different consecutive operations. Another important feature is to provide sufficient space for capacious ore-bins, which are necessary to prevent a stoppage of the mill through a lack of ore, caused through unavoidable delays in the mine or along the roads. The accessibility of the mill site as regards fuel, water, or electrical transmission, according to the motive power to be used, and their continuity and cost at all seasons of the year, must likewise be considered. The possibility of placing the levels for the different floors on solid rock foundations should be investigated, as stability of the machinery is most essential for successful milling.

The ideal site would be to have the mill in close proximity to, but below the level of, the collar of the shaft or the mouth of the tunnel, on sloping ground, where the ore can be delivered directly from the mine to a "grizzly" on the upper floor of the mill, to be passed later, without rehandling, through the crushers, ore-bins, self-feeders, mortars, etc., while leaving sufficient space for a waste dump. For a mill arranged in this manner, including concentrators and canvas platforms, 40′ of fall should be available. If chlorination works are also to be used, a greater fall is desirable.

## MILL CONSTRUCTION.

After deciding on a suitable site, the surface should be removed down to the bedrock and leveled off for the different floors. Solidity and accessibility are the chief points to be observed in placing the different parts of the mill. Where required, heavy stone walls should be erected as buttresses. The foundation for the mortars and the proper erection of the battery frames are points requiring particular attention. For the mortar-block, a trench is prepared of suitable depth, preferably in solid bedrock, proportioned to the height of the block, and wide enough to leave about 2′ of free space around it, which is later filled in with concrete or tailings from the battery. These mortar-blocks vary from 8′ to 15′ in length and are dressed at the upper end to the size of the bed-plate of the mortar. In California they can be obtained frequently from a solid cut of a pine tree, or else consist of two or three sawed blocks fitted and bolted together; but where clear timber of the requisite size is difficult to obtain, the block can be constructed of 2″ plank, as is done in the Black Hills in Dakota.* There the bottom of the trench for the block is leveled and some sand tamped down, on which two layers of 2″ plank are placed crosswise and spiked to each other, and made perfectly horizontal. On this foundation a mortar-block is constructed of 2″ planks, from 11′ to 14′ long, according to the depth of the trench. The planks, which should be of clear lumber, and varying breadths (in order to break joints), stand on end, with their width parallel to the long side of the mortar. They are spiked together and fastened above and below with binders bolted to each other by transverse rods; the upper binders (8″ x 12″) being even with the top of the mortar-blocks; the lower binders (12″ x 12″) are 3′ lower.

The top of the mortar-block should be planed perfectly true and

*Gold Milling in the Black Hills, by H. O. Hoffman. Transactions of the American Institute of Mining Engineers, Vol. 17, 1888–89.

leveled, and where several blocks are placed in line all the blocks should be sawed off to one height. Before setting the mortar upon the block a sheet of rubber cloth, ¼" thick, should be placed between, or when this is not obtainable, two or three folds of mill blankets, well tarred, will answer the purpose.

The mudsills should be of square timber, free from sap, bedded in concrete on the bedrock and secured by anchor bolts to the foundation; they also should be bolted to the linesills.

The uprights of the battery frames are supported in various styles: with diagonal braces and hog chains at front or back, or with so-called knee-frames. In the former style the brace is placed on the same side as the counter-shaft, which rests low down on the battery-sills. This style is well suited for small mills using stamps not to exceed 750 lbs., but for large mills using heavy stamps the knee-frames are the more suitable, with the counter-shaft on a level with the cam-shaft. What is known as the reversed knee-frame forms a strong, compact construction, but requires the counter-shaft to rest on the battery-sills behind the frame. The accompanying cuts show the construction of back and front knee-braces as supplied with mills from the Union Iron Works of San Francisco.

Fig. 2 presents a view of a back-knee battery frame with a Union mortar and square ore-bins, showing the latest arrangement of working the self-feeder from a collar on the stem, instead of having the tappet strike the bumper.

Fig. 3 shows the back-knee battery frame with the cams revolving toward the ore-bin, with a Hayward mortar.

Fig. 4 shows the suspended ore-feeder with back-knee frame.

Fig. 5 is the arrangement of a front-knee battery frame, with Union mortar.

Figs. 6 and 7 show the method of securing knees to the battery posts.

The battery posts are made 24" deep, and from 12" to 20" wide; the center one of a ten-stamp mill being made the heaviest, as having to bear the greatest strain. They are let into the sills and secured to the line timbers by bolts. Besides the braces, the posts are given stability above the mortar by the guide-timbers (see Fig. 6), which extend from end to end in one piece, and are let into the posts to which they are bolted. The lower one is placed about 6" above the upper edge of the mortar, and the center of the upper one is about 3' from the top of the post. The seat for the cam-shaft bearings is cut in the upper part of the posts.

After lowering the mortar on the block, with the planed bottom resting evenly on the sheet of rubber cloth or folds of tarred blanket, it is fastened perfectly rigid by eight bolts, four on each long side, passing through the flange, which is cast on the bottom of the mortar. This flange is 4" wide and about 2½" thick. The feed floor should be high enough so that these bolts can be conveniently reached, to permit their tightening when required. The journals for the cam-shaft, which are placed in the recesses cut out of the battery posts for their reception, are lined up and "babbitted" prior to receiving the cam-shaft with its cams. The stems are placed from ½" to 1" from the cam-shaft, and just far enough from the cams to clear them when dropping. The cam-shaft is made of wrought iron or soft steel, from 4½" to 5" in diameter, turned true, and should have key-seats for securing the cams. There should

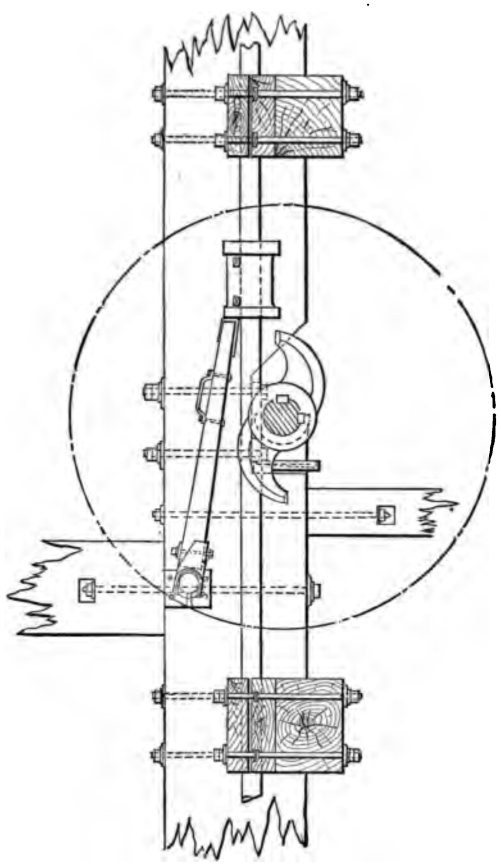

FIG. 6. METHOD OF SECURING KNEES TO BATTERY POSTS.

be two key-seats, and placed one third of the shaft circumference apart. At one end of the cam-shaft the cast-iron "hub" of the belt pulley (with flanges) is keyed on. This pulley is built of wood, and turned true on the shaft. Where there is more than one battery to the mill, it is best to have a cam-shaft for each ten stamps, as this permits of repairs, such as changing cams, etc., without stopping more than ten stamps.

The guides (see Figs. 6 and 7, and D, E, and F of Fig. 8), which direct the drop of the stems, are in two sets, upper and lower—the former above the tappets, and the latter below the cams—and are bolted to the guide-girts by eight bolts. They are best made of hard wood, but pine answers sufficiently well—though the former lasts five times as long as pine. The old style guide (F, of Fig. 8) consists of two pieces of 4″ plank 14″ wide, planed on all sides, and of sufficient length to fit easily between the battery posts, with equi-distant semi-circular grooves, fitting together, for the passage of the stems. A quick and exact way to make these grooves is to clamp the two planed planks tightly together as they are to be placed on the frame, set them on edge, and, after marking off the centers for the five stems, bore out the circle

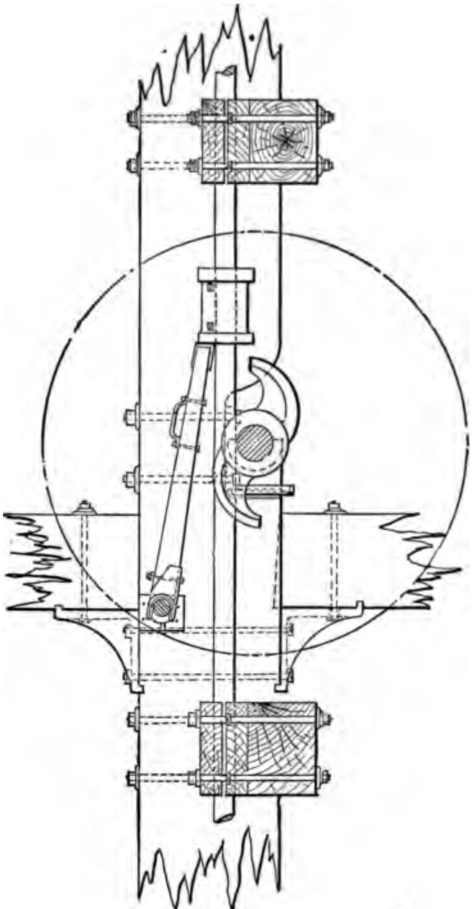

FIG. 7. METHOD OF SECURING KNEES TO BATTERY POSTS.

(using the joint line for the point of center) with a long-handled auger having an adjustable bit. These are kept in many mills for this special purpose.

Before bolting the guides in place, half-inch pieces are placed between the two halves, and adjoining each stem, which are planed down later as the guides wear, leaving but little play for the stem. After boring out the grooves for the stems, and before putting the guides in place, they should be lubricated. A convenient and economical plan is to cut some semi-circular pieces of thin sheet iron of a somewhat larger diameter than the grooves, and drive them into the wood at both ends of the channel; then lay the halves level, groove side up, and fill the latter with linseed oil, letting them remain until the wood has taken up all it will absorb, when the remainder is returned to the can, and the sheet-iron pieces removed.

If, in a pine guide, those portions occupied by the grooves are cut out square, and hard-wood bushings fitted in before boring out, the stem will work parallel to the wood fiber, which reduces the friction,

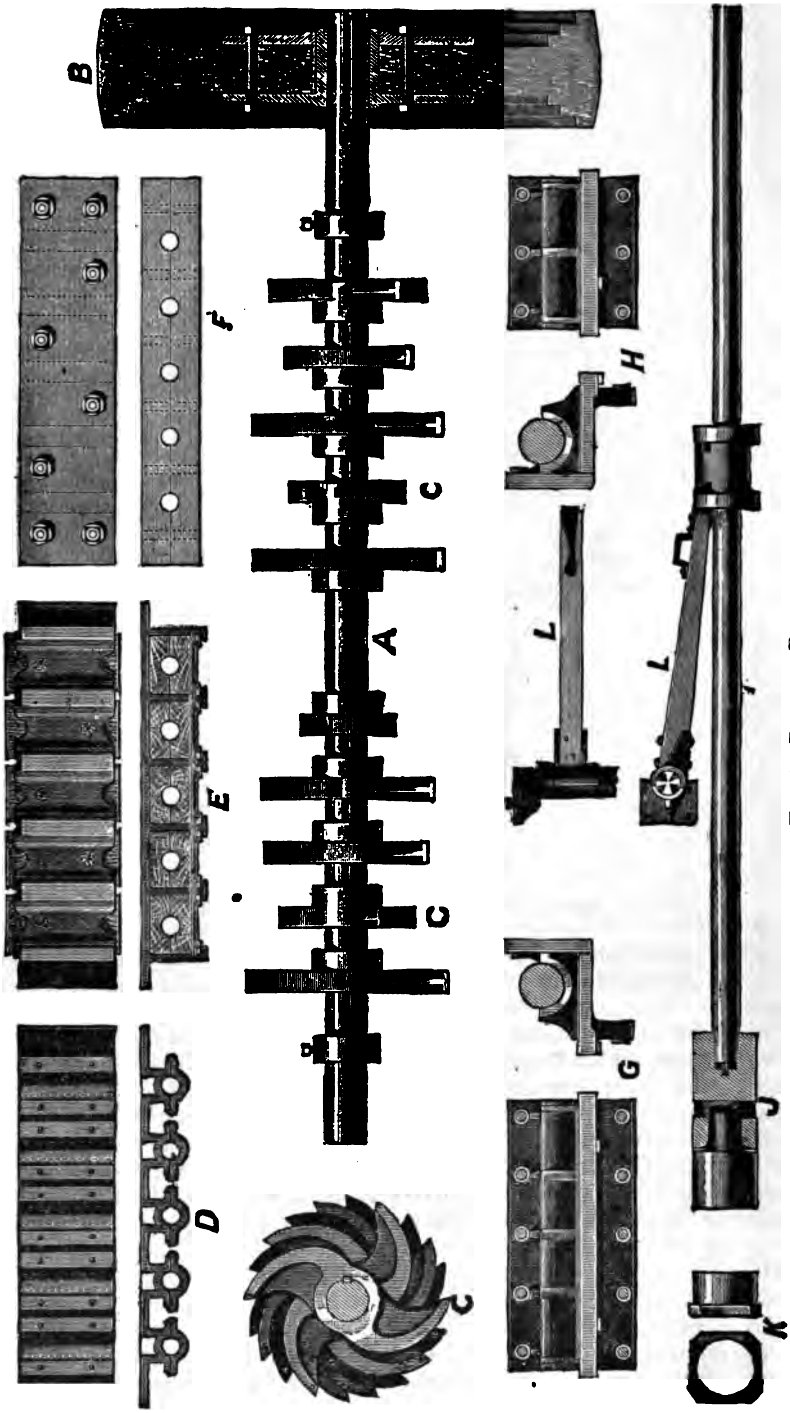


FIG. 8. BATTERY DETAILS.

and lengthens the life of the guides; while only the hard-wood bushings would need replacing, making the cost of the guides less.

The great drawback to these guides is that when a stem has to be removed, the entire battery has to stop; hence the adoption of separate guides for each stem, being either all iron (see D, Fig. 8), or wooden bushing in iron frames (see E, Fig. 8), which are held in place by wedges and the lips of the iron frames.

For the support of the stamp-stems when suspended, wooden latch-fingers, or jacks, are supplied. (See Figs. 6 and 7, and L, of Fig. 8.)

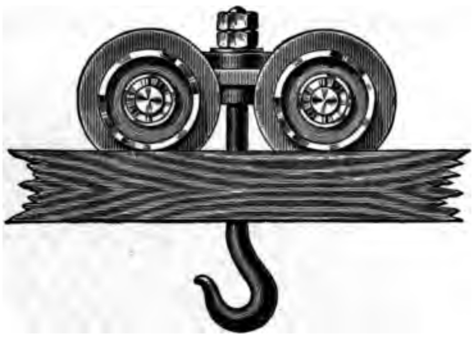

FIG. 9. OVERHEAD TRAVELING CRABS.

A jack-shaft, 3″ in diameter, rests in bearings attached to the inner sides of the battery posts; on this, cup-shaped sockets ride, in which the wooden fingers are attached, shod at the upper end with an iron plate $\frac{1}{8}$″ thick, and provided with an iron or leather "hand-hold" near the top.

For the greater convenience of quickly removing and replacing stems, or cam-shafts, large mills are supplied with overhead travelers, or "crabs," in line with the batteries, in connection with a chain block and tackle running on plates secured to the roof, shod with iron tracking. To easily reach the cams, tappets, etc., a platform is placed just below the cam-shaft.

The feed-floor consists of a double board floor of 1″ lumber, with broken joints, supported on joists 18″ apart, and about 2′ below the feed opening in the mortar.

MILL DETAILS.

*The Grizzly* is a coarse screen consisting of a number of parallel bars attached to a frame, set on an angle from 45° to 55°, over the ore-bin. These bars may be of round, rectangular, or V-shaped (apex down) iron, or of wood, faced with iron, and resting on several iron cross-rods, held apart with iron washers; the distance between the bars should be equal to the opening the rock-crusher jaws are set to—from 2″ to 3″. There are no fixed dimensions of length or breadth, as these depend in a measure on local conditions; but they are usually from 3′ to 6′ wide, and long enough (12′ to 15′) to give the fine material time to drop through the spaces before reaching the crusher floor.

Where substantial steel T rails are used for tracking in the mine, they can be made to serve for grizzly-bars when no longer of use in the mine, by turning them with the base up.

The grizzly should be placed at the highest point of the mill over the ore-bin, where the car or wagon can enter and dump. Its chief object is to separate at once the finely divided ore from the coarser; a secondary purpose is served in affording an opportunity to recover drills, gads, or hammers that may have come from the mine, in the ore, before they reach the rock-breaker or mortar. Its lower end rests on a platform in front of the rock-crusher, or better, in a chute with an adjustable end-gate placed above the mouth of the rock-crusher so as to permit of its being fed automatically.

Where the ore as delivered from the mine carries less than 5% of fine stuff, the grizzly should be dispensed with, especially where, in con-

FIG. 10. GRIZZLIES.

structing the mill, fall must be economized. Some object to the use of the grizzly, as tending to feed all the hard rock by itself, and say the output of those batteries is below the others.

*Rock-Breakers or Crushers* are placed on a platform below the grizzly and above the ore-bin in such a manner that the crushed rock mingles with the fine stuff passing between the bars of the grizzly. The rock-breaker must be of sufficient weight to remain firm in its place, and strong enough to resist heavy strains; the dies should be easy to exchange and adjust, and all parts requiring to be oiled should be arranged to prevent oil coming in contact with the quartz. In large mills it is best to have one crusher to supply every twenty stamps, and on account of their intermittent work, they should have driving power separate from that of the stamps.

Rock-breakers are adjusted to crush the rock smaller than the throat of the mortar (therefore, less than 3″), but as the work of the rock-breaker is cheaper than that of the stamp, it would pay, with very hard rock, to do more of the crushing with this machine, even to the extent of placing two crushers, one beneath the other, and bringing the quartz greatly reduced to the stamps.

There are two general types of rock-crushers. The older pattern

carries a flat, fixed jaw, working with one having a reciprocating motion and using flat or corrugated dies that are reversible. The Blake is representative of this pattern. The other pattern has an outer, circular, fixed jaw, within which a corrugated jaw circles, of which the Gates is representative. This latter machine permits of larger blocks being fed. It is an excellent machine for heavy work, and where the rock is not wet or clayey; but it requires greater horse-power, for where a Blake, 10″ by 8″, crushing 3 tons per hour, requires 9 H. P., the Gates, with a diameter of 37½″, crushing 3½ tons per hour, requires 16 H. P. The Gates consists of a nearly vertical shaft of forged steel, rotated from below by a

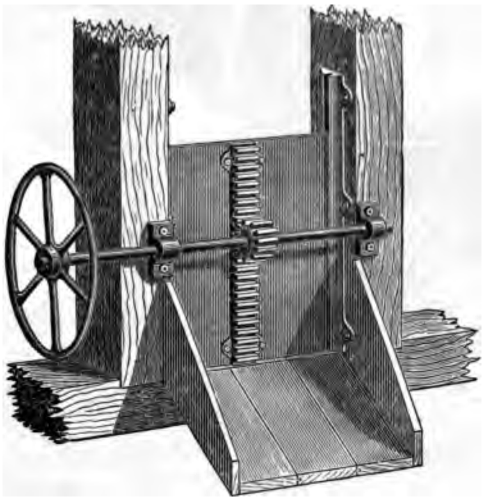

FIG. 14. ORE-BIN GATE.

beveled wheel set ½″ out of center, on the top of which a chilled-iron conical head is attached, with the base downward, rotating within chilled-iron concaves, with an outward slope, set in the cylindrical body of the machine. Between these two faces the ore is crushed, their distance apart below being gauged by set-screws. The shaft, by being made to revolve around an eccentric at the bottom, has a constant crushing power without doing any grinding. A set of concaves lasts two years, and can be replaced; the center shaft with the chilled-iron head has been known to crush 120,000 tons of an average hard quartz before wearing out.

*Ore-bins* should always be as spacious as the surroundings will permit, but never of less capacity than will carry a twenty-four hours supply for the mill, say about 65 cu. ft. to the stamp. They are usually constructed with a sloping bottom, to facilitate discharging, but where very large bins can be erected this feature is not so essential. These bottoms must be solidly braced and ought to be covered with iron plates over those portions where the ore has to be dropped. The front of the bin is parallel with the mortars and supplied with gates for each battery above the level of the hopper of the self-feeders. These gates should be regulated by a pinion and rack, and set for a regular discharge and delivery, through chutes, into the self-feeders. The chutes should be lined with heavy sheet iron.

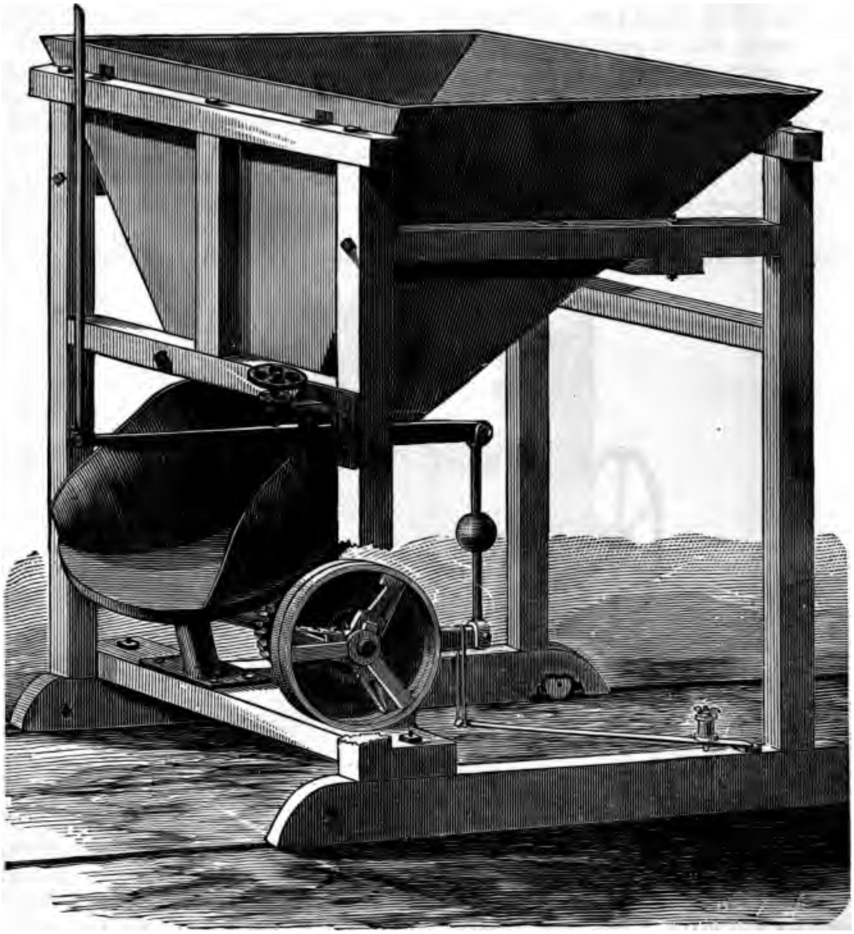

FIG. 15. HENDY CHALLENGE ORE-FEEDER.

*Self-Feeders.*—The entire value of the stamp battery hinges on a regular and even feeding, and as it can be done much better (from 15% to 20%) by a machine than by hand, this latter method has become well nigh extinct in California. Among the mechanical feeders mostly used are the Challenge (in two patterns), Tulloch, Stanford, and Roller feeders. Although the three latter are very serviceable for certain classes of ore, and are cheaper in first cost, the Challenge is undoubtedly the best all-round machine, which is proved by its almost universal adoption. They are either placed on a frame which runs on an iron track in the feed floor, back of and at right angles to the battery, or are suspended from tracks supported by the battery posts and standards placed against the ore-bin. This latter pattern permits of greater accessibility to the feed side of the mortar. In general, the Challenge feeders consist of a hopper with a movable circular plate beneath, set slightly inclined toward the mortar, receiving a rotary motion by means of gear wheels acting on the lower face of the plate, which are moved

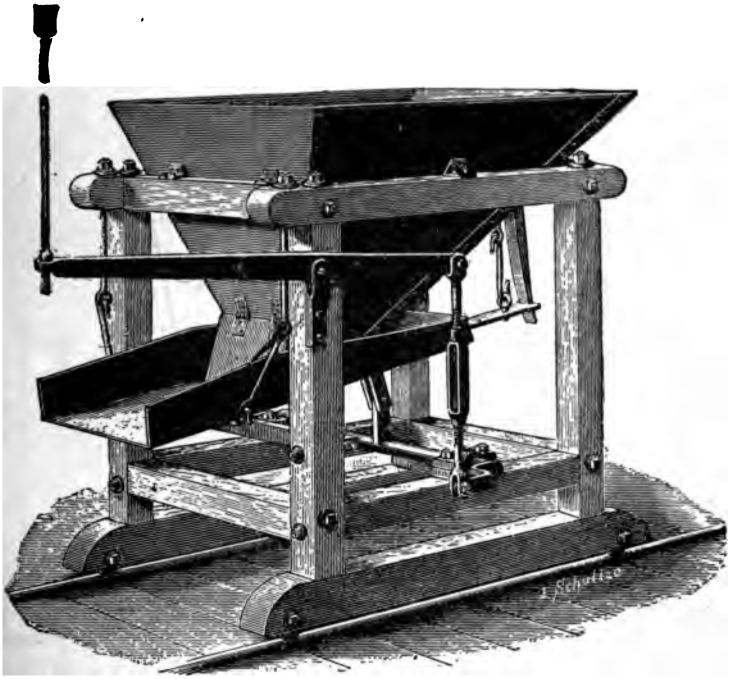

FIG. 16. TULLOCH AUTOMATIC ORE-FEEDER.

by a friction grip that receives its impetus from a blow of the descending stem on a bumper-rod connected with it. Movable wings extending from the point of the hopper over the plate toward the throat of the mortar permit a given quantity of the ore to be scraped off at each blow through a partial rotation of the plate. The older machines were made with right or left-handed bumpers, but the present and better plan is to place the rod in the center, so that the third stamp in a five-stamp battery imparts the blow. The newest machines have no bumper-rod, but are worked by a collar fastened on the stem above the top of the mortar. Each battery is supplied with its own self-feeder.

The Tulloch feeder consists of a square frame, into which a hopper fits, having below a tray suspended from the frame at any desired angle, and in such a manner as to have a forward and backward swinging motion inside the frame, which can be arrested on the forward motion at a certain point by lugs, underneath the tray, striking a bar. The back of the hopper is supplied with an adjustable scraper, and at each motion of the tray a certain amount of the ore is scraped forward and falls into the battery. The machine is operated by the descent of the stamp.

*Mortars.*—The mortars in California are mostly single-discharge, and cast in one piece, extremely solid. When required in places inaccessible by wagon roads, they are cast in pieces, which are later bolted together. Their interior form depends on the nature of the ore, and the procedure to be applied; thus we find them made with narrow or flaring, deep or

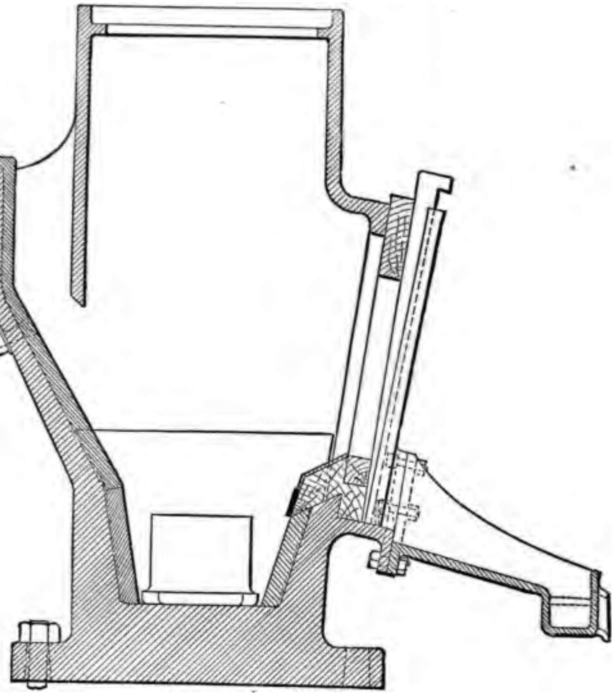

FIG. 17. HAYWARD MORTAR.

shallow troughs, and with or without inside plates. Mortars with narrow troughs are made for greater output, while a wide trough assists battery amalgamation, and gives opportunity for placing inside copper plates. In some of the newest styles of mortars a series of grooves are furnished in the lining plates, to contain quicksilver. The mortars weigh from 4,000 to 6,500 lbs., the bottoms being made extra heavy; in some of the latest patterns the bottoms are 8″ thick. The length varies between 4½′ and 5′, and the height from 4¼′ to 4½′. The inside width of the trough corresponds with that of the foot-plate of the dies. A heavy flange, 4″ x 3″, is cast on the base of the long sides, in which are four holes on each side for bolts, to secure the mortar to the block.

The difference in design hinges chiefly on the different opinions of leading mining men as to the method and value of amalgamating inside the battery.

Figure 17, known as the Hayward mortar, is a full-lined mortar, with flaring trough, weighing, complete, about 6,500 lbs., without any special arrangements for inside amalgamation.

Figure 18, the Alaska mortar, is a full-lined mortar, with flaring trough, in which the linings are furnished with grooves, to contain quicksilver.

Figure 19, the Wilman's mortar, was the first attempt at inside amalgamation, using an inside removable copper plate, but this failed to work well; the copper plate, being so close to the shoe and die, scoured, and could not retain the amalgam. In remedying this defect, the

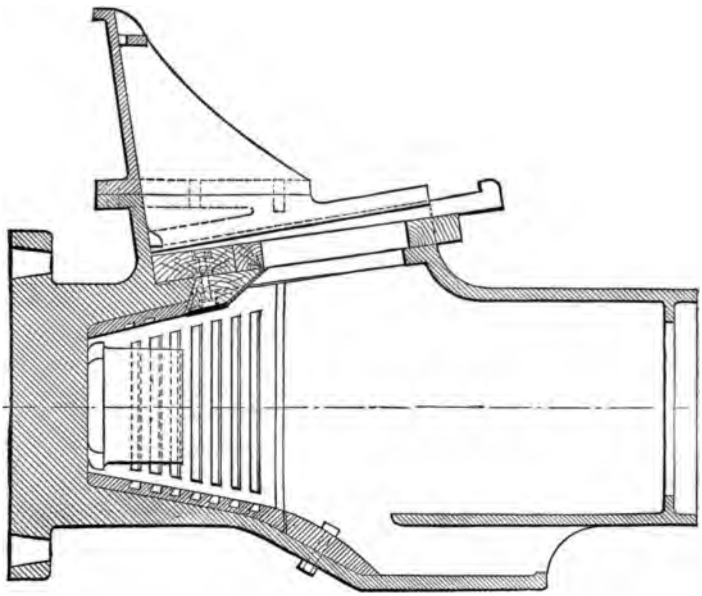

FIG. 18. ALASKA MORTAR.

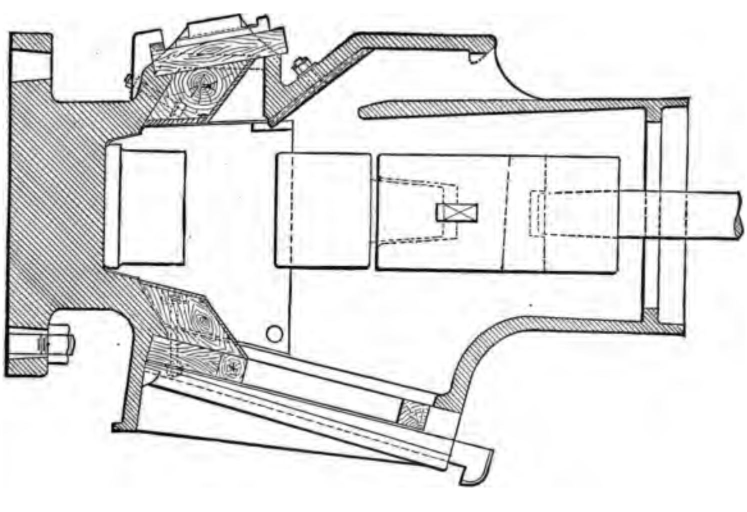

FIG. 19. WILMAN'S MORTAR.

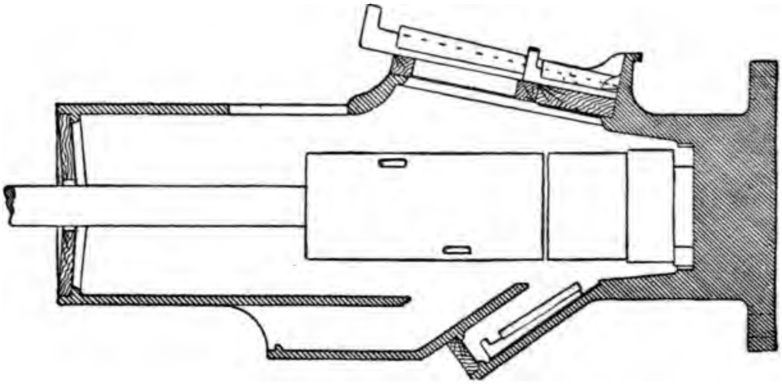

FIG. 22.

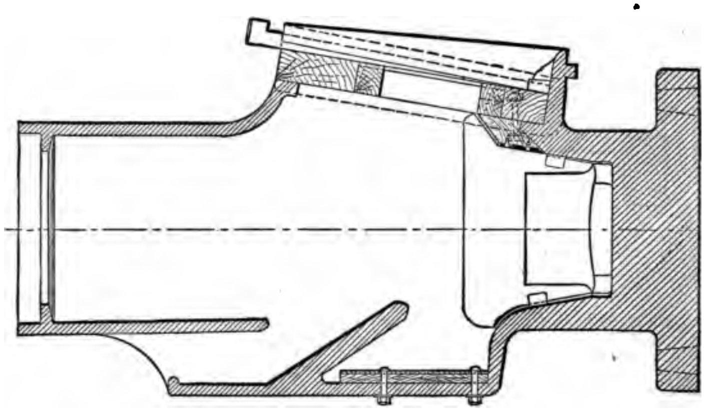

FIG. 21. UNION MORTAR.

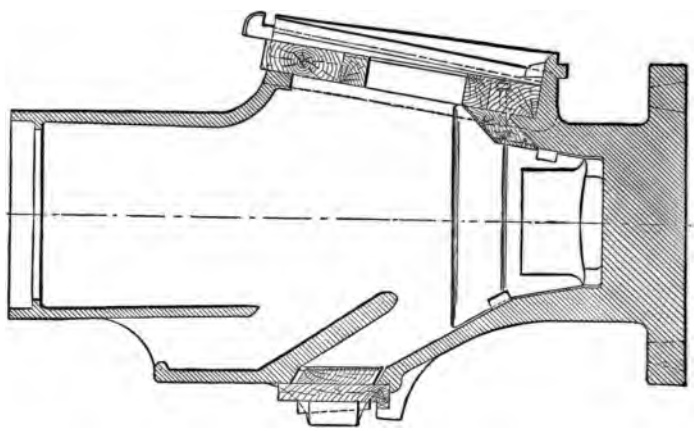

FIG. 20. PACIFIC MORTAR.

Pacific mortar (Fig. 20) was produced, in which the copper plate was placed higher up, under the feed throat; experience in the working of which, suggested changes that finally evolved the Union mortar (Fig. 21), which is provided with a copper amalgamating plate, 12"x48", bolted in a recess at the back of the mortar, reaching below the level of the screen opening. This plate is removed on clean-up days to be scraped, and then replaced. Similar good results have been obtained by the writer in using a double-discharge mortar, and filling the back discharge opening with a plank instead of a screen, to which a plate, 8" wide and the length of the opening, had been attached.

A modification of this back-plate arrangement is shown in the accompanying drawing (Fig. 22) of a mortar designed in Milwaukee, Wis. Here the back plate is accessible, from the back of the mortar, through a covered opening; it is secured in place by a dovetailed key at each end, allowing it to be adjusted to the varying height of the dies.

*Dies* (see K, Fig. 8, and S, Fig. 23).—They consist of a cylindrical body of the same diameter as the shoe, with a square foot plate with broken corners, and should fit loosely against the front and back plates of the mortar. The broken corners permit their easy removal. They are cast both in iron and steel.

*Shoes* (see J, Fig. 8, and P, Fig. 23).—They are made of iron or steel, and consist of a cylindrical body of the same diameter as the stamp-head, with a cone-shaped neck, half as wide as the cylindrical body, and about 5" long. The weight of the shoe bears a certain relation to the other parts of the stamp, generally about one sixth of the total weight when made of chrome steel, but somewhat less when made of iron. The cylindrical portion of the shoe is somewhat longer than the corresponding part of the die, on account of its greater wear—the latter being protected by a cushion of quartz. Both shoe and die are used until worn as thin as possible; with the shoe, this may be ½", though rarely, while the die is worn to the foot-plate, if not fractured previously. This practice is not to be commended, and should only occur in case of necessity. On stamps weighing about 900 lbs. the shoes, if of chrome steel, weigh about 150 lbs., and if of iron, weigh about 20 lbs. less, and are about 9" in diameter. The life of the shoes depends on the nature of the quartz and the height and speed of the drop, but as a general rule shoes and dies of steel last as long as two and a half sets of iron ones, and cost twice as much. In the matter of choice between steel and iron, the vicinity of the mill to foundries is of consequence. Steel shoes and good iron dies usually work very smooth, but where the waste iron can be disposed of at a foundry, this metal is preferred for both.

*Stamp-Heads, Bosses, or Sockets* (see J, Fig. 8, and R, Fig. 23).—They are made of cast iron or steel, of the same diameter as the cylindrical part of the shoes and dies, with two conical sockets; the upper one accurately bored out to contain the tapering end of the stem, and the lower one to receive the neck or shank of the shoe with its inclosing circle of thin wooden wedges. Transverse, rectangular keyways, at right angles to each other, pass through the stamp-head at the end of the conical openings, connecting therewith in such a manner that when both stem and shoe are attached to the boss they protrude into the keyways. This enables them to be forced out by the driving in of a wedge-shaped

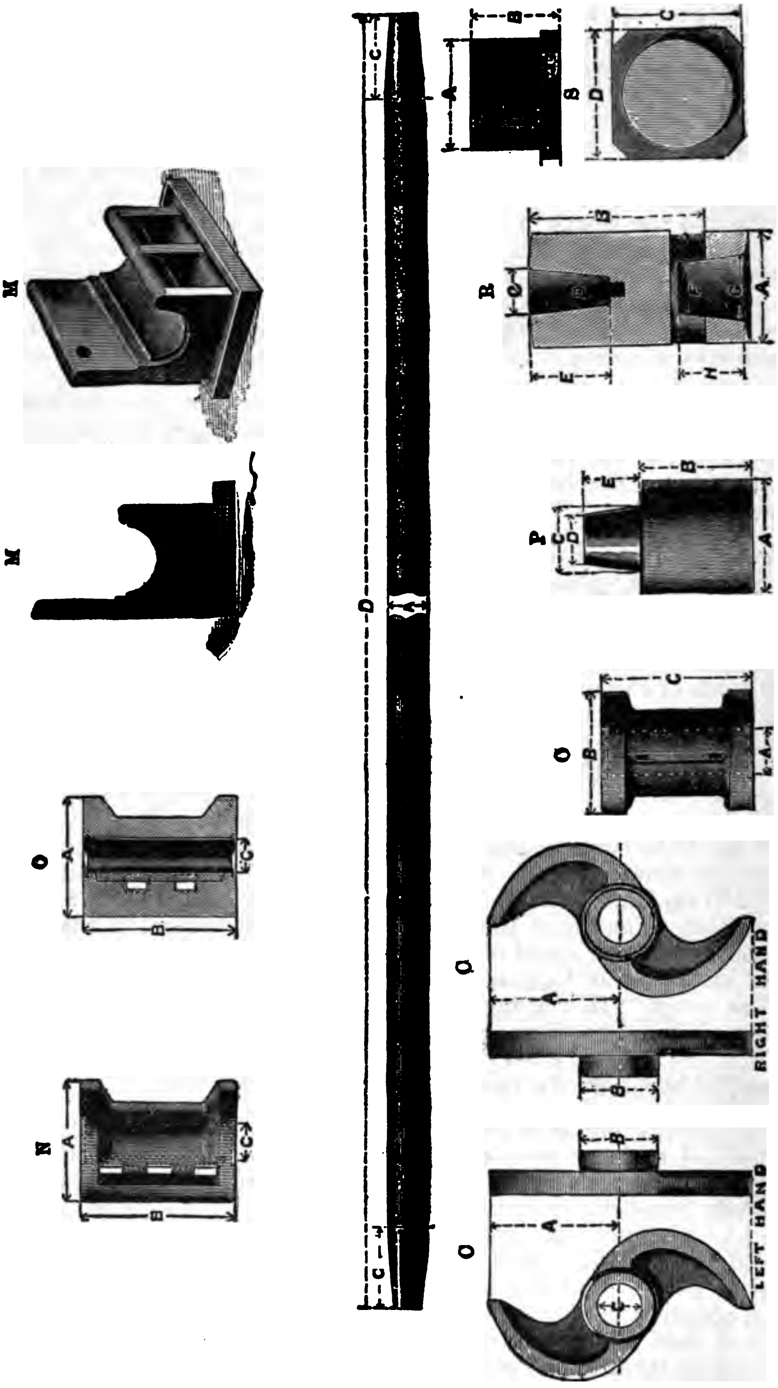


Fig. 23. Battery Details.

steel drift about 1″ wide, 18″ long, and tapering down from 2″ toward the point. The ends of the stamp-head are usually reinforced by having iron bands shrunk onto them.

*Stems* (see I, Fig. 8, and D, Fig. 28).—They are made of wrought iron or soft steel, turned perfectly true, and tapered at both ends for a distance of 6″ or 8″. They are from 11′ to 14′ in length, the diameter varying with the weight of the stamp from 2⅞″ to 3½″. They are reversible, so that if one end breaks, the other end can be used before sending to the shop for repairs. When this repair is made, the whole stem should be annealed. The stem carries the greater weight of any part of the stamp, amounting to nearly one half. The stems hang in the guides at even distances from center to center, and are supported while at rest by props or fingers catching on the under face of the tappets.

*Tappets* (see O, N, and O, Fig. 23).—They are made of tough iron or steel, cylindrical, with a flange on both ends, and accurately bored through the center, a shade wider than the stem, and counter-bored at both ends; they are provided with a rectangular recess adjoining the central bored hole, 7″ to 8″ long, and from 2″ to 2½″ wide, in which a gib is fitted. This is a piece of wrought iron or steel, grooved on one side, with curvature ⅛″ smaller than that of the stem, and planed flat on the opposite side. Two, or in some cases three, slots are cut through the tappet between the flanges, at right angles to the stem, which connect with the rectangular recess for the gib, so that keys, when driven through the slot, press the gib against the stem, which should slide smoothly through the center of the tappet. The tappets are faced on both ends, and are reversible. The keys are of steel, fitted and marked. Tappets weigh from 100 to 120 lbs. When fastening the tappet, the keys are driven in solid; but care must be observed, as when too tightly keyed, the tappet is liable to split.

*Cams* (see C, C, Figs. 8 and 23).—They are of tough cast iron or steel, double armed, and strengthened by a hub; which latter is frequently reinforced by having a wrought-iron ring shrunk on. The cam itself is the involute of a circle having for its radius the distance between the center of the stem and the cam-shaft, somewhat flattened, however, at the point of the cam. It comprises a face from 2″ to 3″ wide, ground off, decreasing in thickness from the hub to the point, and strengthened by a rib on the under side, which runs from a point to several inches deep at the hub. The cam is fastened to the cam-shaft by steel, hand-fitted keys.

*Cam-Shafts* (see A, Fig. 8).—They are made of wrought iron or soft steel, turned true, with double key-seats, 120° apart for cams, besides key-seats for the driving pulley. The cams are slipped on the cam-shaft with the hub side away from the stem, and keyed solidly in their respective places; they must be placed in such a manner that when the cams are raising the stamps, the weight is as nearly evenly distributed over the shaft as possible. For this reason proper attention must be given to the sequence in which the stamps are to drop in the battery. Where the shaft is for ten cams, the following order or succession of drops is

recommended, viz.: 1, 5, 9, 3, 7—10, 6, 2, 8, 4, and would give a drop in each battery as follows: 2, 4, 1, 3, 5.

The cam, in picking up a stem on the under side of the tappet, imparts a revolving motion to tappet and stem, requiring from four to six strokes of the cam to complete one entire revolution. A too rapid revolution indicates the need of lubricating. The revolving of the stem assists in giving an even wear to the faces of the shoes and dies, but it does not impart a grinding action to the stamp, as frequently stated, which can be proved by holding a piece of chalk against the stem during its ascent and descent.

*Screens and Frames.*—Screens of different materials and with different orifices are used; the materials comprise wire cloth of brass or steel, tough Russian sheet iron, English tinned plate, and, quite recently, aluminum bronze. The Russian sheet-iron plates are perforated with round holes or slots; the latter are vertical, horizontal, diagonal, or curved, and are either entirely smooth or burred on the inner side. The latter form is intended for longer wear by closing the burrs with a mallet when too large, thereby prolonging the life of the screen. These screens last from fifteen to thirty-five days. The plates have glossy, planished surfaces, and come in sheets of 28″ to 56″, costing in San Francisco from 65 to 80 cents per square foot. The English tinned-plate screens come in sheets of 1′ to 1½′ square; they are more flexible than the Russian iron, hence do not permit of the pulp caking along the lower edge when fed high; and, as compared with a Russian iron one of the same perforations, they give a greater discharge, but they are short lived—averaging about ten days. The tin is burned off before using. Brass screens, costing in San Francisco 36 cents a square foot, are sold in rolls; they give the greatest discharge for an equal area, and last from ten days to two weeks, but should not be used if cyanide of potassium be used in the battery, on account of clogging with amalgam. The "aluminum bronze" plates come in sizes similar to the sheet tinned plate, but unpunched, the latter work being done here; they are much longer lived than either of the other kinds, and have the further advantage that when worn out they can be sold for the value of the metal for remelting; these plates are bought and sold by the pound, and are said to contain 95% of copper and 5% of aluminum. Steel-wire screens are not much used, on account of their liability to rust. The life of a screen depends, aside from the manner of feeding, on the width of the mortar, the height of the discharge, and the hardness of the rock. Wide mortar and high discharge are favorable to the preservation of a screen; the form of the perforations—round holes, or slots, etc.—influences the discharge area of the screen.

A good deal of confusion exists in interpreting the numbers of the different kind of screens. Wire screens take their numbers from the meshes to the linear inch, while perforated and slotted screens are numbered from the needle used in punching them, these needle numbers being the same as are used for sewing-machines. The sizes most frequently used in gold milling are from No. 6 to No. 9 of the perforated and slotted screens, and from No. 30 to No. 40 of the wire screens. The slots are from ¼″ to ½″ long, and placed alternating or even in the rows, some being burred on the inner side.

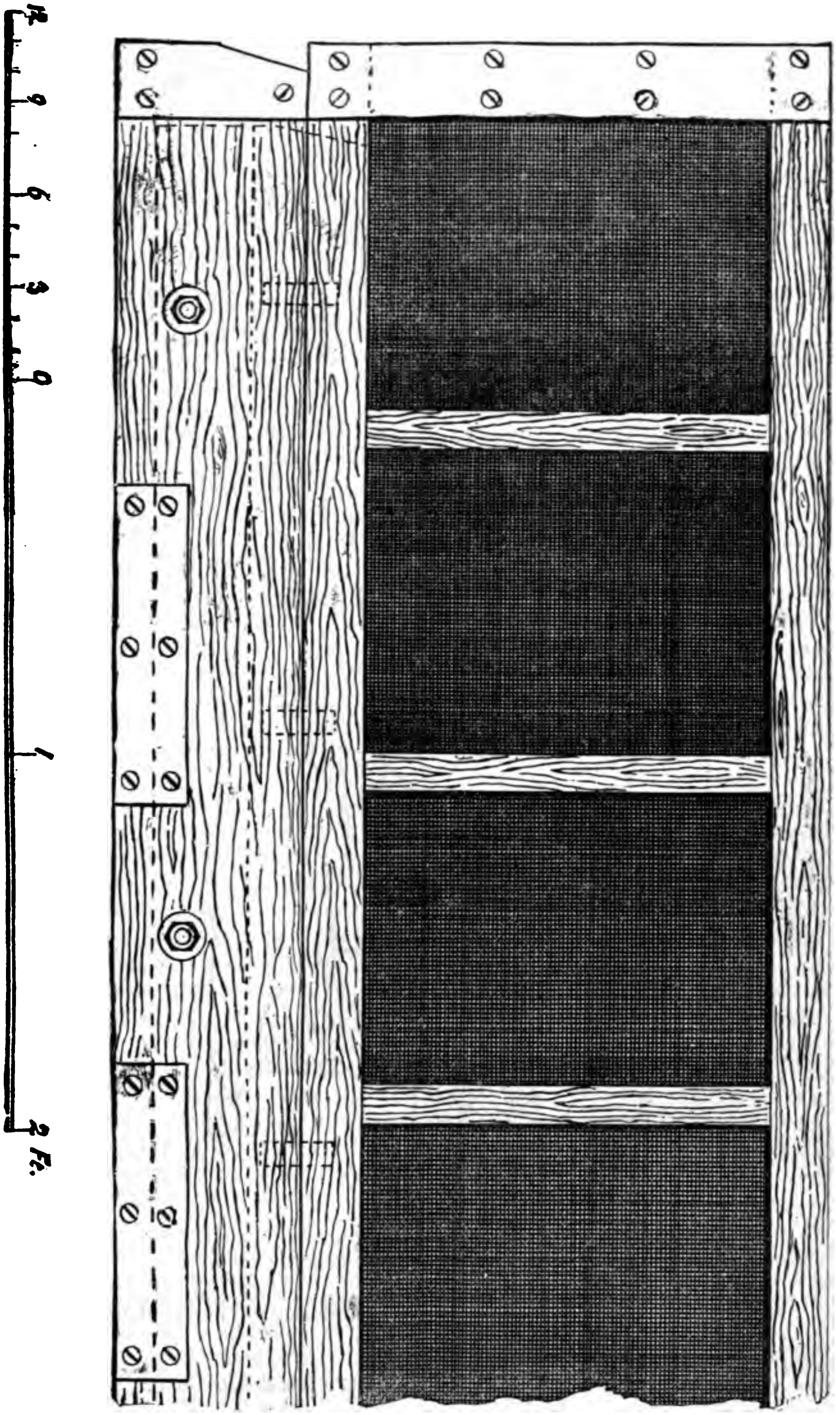


FIG. 24. BATTERY SCREEN FRAME.

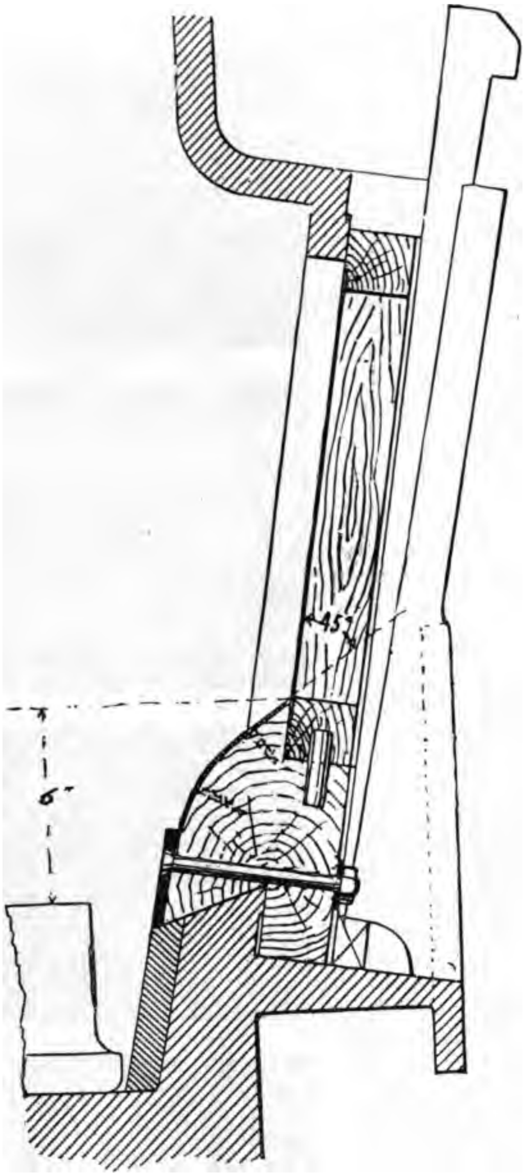


FIG. 25. ADJUSTABLE BATTERY SCREEN.

As dies wear down, wooden chock-blocks (on which the inside plates are fixed), of les height, are substituted, thereby preserving uniformity in height of discharge.

The following table gives a comparison of the different varieties, with their numbers:

| No. of Needle. | Corresponding Mesh. | Width of Slot. (Inches.) | Weight per Square Foot. |
|---|---|---|---|
| 5 | 20 | 0.029 | 1.15 lbs. |
| 6 | 25 | 0.027 | 1.08 lbs. |
| 7 | 30 | 0.024 | 0.987 lbs. |
| 8 | 35 | 0.022 | 0.918 lbs. |
| 9 | 40 | 0.020 | 0.827 lbs. |
| 10 | 50 | 0.018 | 0.735 lbs. |
| 11 | 55 | 0.016 | 0.666 lbs. |
| 12 | 60 | 0.015 | 0.666 lbs. |

The proper size required is a matter for the millman to decide at each mill. The character of the ore and the coarseness of the gold have to be considered, as well as the inside dimensions of the mortar; ore carrying extremely fine gold requiring a finer crushing, as the gold must be freed from the quartz matrix in part if the quicksilver is to act on it; but where this would lead to if carried out to its legitimate end may be imagined, when the writer states that he has observed, under the microscope, a particle of quartz that had passed through a No. 9 screen (40-mesh) and still contained several separate but included particles of gold. Sulphide ores, having a much greater tendency to form slimes, should be crushed as coarse as permissible, and where the sulphides predominate largely, amalgamation in the battery is best avoided. The pulp discharged through a screen carries but a small percentage of the size of the orifice; while the largest proportion is much finer, it is possible to use a much coarser screen than the size desired to be obtained without any great detriment, while greatly increasing the output.

*The Screen Frame* (Figs. 24 and 25).—It is made from strips of sugar pine 1½" by 3" broad, mortised, and reversible; usually they are made to close the entire discharge opening, grooves being cast on the exterior of the mortar for their reception. It is frequently strengthened with one or more vertical ribs across the center opening, and is faced with iron plate on those portions of the side and bottom that come in contact with the iron keys that hold the frame solid against the mortar. In some mills the frame is made several inches lower than the opening, to permit the millmen to observe the interior of the mortar while in action, and to allow the hand to be introduced to remove any chips that may have passed in with the ore, as these have a tendency to bank up against the screen and interfere with the discharge of the pulp. Where such a screen frame is used the opening above is kept covered with a strip of canvas tacked to a wooden rod, laid on the upper projecting lid, while the loose end of the canvas hangs against the inside of the upper part of the screen frame.

*The Plate-Block* (*Chock-Block*) consists of wooden blocks bolted solidly together, and fitted and keyed to the lower edge of the mortar along the discharge opening, with one part projecting above the other, forming a recess on top to contain the screen frame, and lined with a piece of blanket to make a close joint. The inner side is sloped or rounded off, and fitted with an amalgamated plate. The front and ends are faced

with iron plate to protect the wood. Two or more sets of these chock-blocks should be provided, of which one stands 2″ higher than the other; they are then used alternately, the higher one with new shoes and dies, to be replaced by the lower one when the dies are worn down somewhat, to retain a more even discharge than would otherwise be possible.

*The Drop* is the height through which the stamp is raised by the cam, and through which it drops when released. Usually it is the same for all the stamps in a battery, although the end and feed stamps sometimes receive a different drop. It is regulated through the raising or lowering of the tappet, and depends mostly on the hardness of the rock. It is one of the factors in determining the speed with which the blows from the stamp shall be repeated. The usual combination of the two in the California mills, is a low drop with rapid motion.

*The Discharge* is the distance between the top of the die when in place in the mortar, and the lower edge of the screen through which the pulp discharges. It is one of the most important factors in the duty of the stamps and the gold output from the ore. It should be maintained as nearly as possible at an even height through the entire period of crushing; the height of the chock-block or screen frame being lowered to correspond with the wear of the die. A further means used to retain an even discharge is by placing a 2″ iron plate under the dies, when worn thin. The discharge stands in a certain relationship with the fineness of the screen: low discharge goes with coarser crushing, a high discharge with the opposite. The discharge varies in California from 4″ to 10″.

*Water Supply.*—Water-pipes of 3″ diameter are brought along the front of the mortar near the upper edge, with branch pipes 1″ in diameter, supplied with faucets leading to the feed side of the mortar, to convey the battery water in at the back, or through the plank-covering on the top; this water is under moderate pressure. A second discharge pipe is carried down in front to the lower lip of the mortar, where a movable, perforated branch is turned across the front of the screen, discharging along the entire line on the lip; this second discharge pipe also supplies a hose. The battery water should enter both sides of the mortar in an even quantity, and the total amount must be sufficient to keep a fairly thick pulp that discharges freely through the screen. About 120 cu. ft. of water per ton of crushed ore may be considered an average, or 8 to 10 cu. ft. per stamp per hour.

*Aprons and Apron-Plates.*—The apron is a low table placed in front of the mortar, just below and in immediate proximity to the lower lip of the discharge, for the reception of amalgamated copper plates. It is set on a sufficient grade to permit the discharging pulp to flow over it in an even stream, while affording the suspended amalgam an opportunity to reach, and adhere to, the plate surface. The size, shape, and slope are at the will of the millman; but usually they are rectangular, with the plates screwed down to the table with copper screws, perfectly level and smooth, the sides being secured with wooden cleats. The grade given varies from ½″ to 2½″ to the foot, and the width of the apron is usually the width of the discharge-opening of the mortar. In some

mills several of these apron-plates are placed consecutively, discharging from one to the other. They are usually rigid, but in some instances the apron next to the mortar stands on rollers, permitting it to be rolled back, and thus giving freer access to the front of the mortar. They should not be attached to the battery frame.

*Sluices and Sluice-Plates.*—These vary from 12″ to 20″ in width, and are placed below the aprons; they are usually set to a grade different from that of the apron. The plates can be fastened by cleats, or are laid overlapping at the ends, and, if not wider than 16″, do not need to be fastened down with side cleats; this permits of their being picked up and cleaned at any time without stopping the battery. The sluices are rarely over 16′ long—more frequently in lengths of 8′—and should always be placed double. The width and grade, as compared with the apron areas, are mostly faulty in California mills.

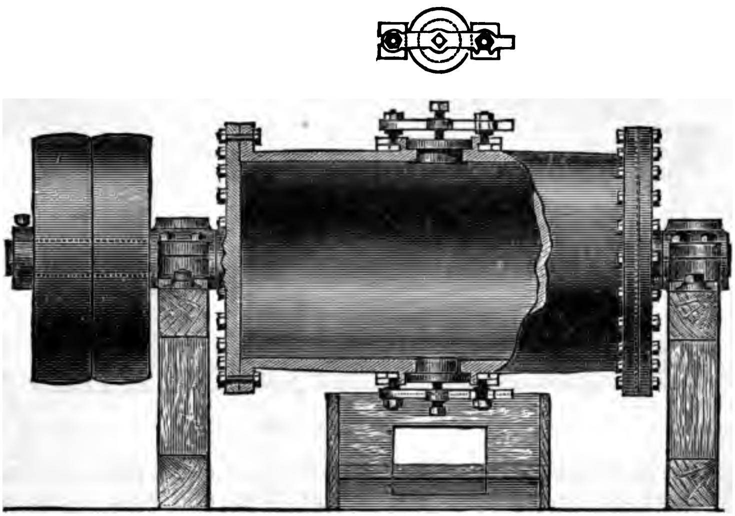

FIG. 26. CLEAN-UP BARREL.

*Clean-up Barrel.*—Large mills are supplied with clean-up barrels, which consist of iron barrels supported by trunnions resting in bearings on short standards. One of the trunnions is extended to carry a loose and a tight pulley, by means of which it is revolved. A manhole, with tight-fitting cover, is provided for charging and discharging, and below it is a sluice with cross-riffles to receive the pulp when discharged from the barrel. The barrel should make from thirty to forty revolutions a minute, requiring 2½ H. P. It is used to treat the battery sands when cleaning up the mill; also, all the scrapings from the mill floors, as well as sand from the drop boxes and amalgam traps, large pieces of quartz or pieces of broken shoes being added with water and quicksilver to assist in the operation.

*Clean-up Pan* (Fig. 27).—This is a small amalgamating pan, 3′ to 4′ in diameter, operating with mullers with wooden shoes, and is run at a speed of thirty revolutions, requiring 1½ H. P. When in use the pan is half filled with water, and the amalgam put in, with an addition of clean

quicksilver, and, if required, also some lye. After sufficient grinding, the muddy water is run out through plug-holes, the mullers stopped, and the contents drawn off in buckets. The iron found floating on top of the quicksilver is removed with a magnet; the sand is washed off with a small stream of clear water, and if any dross be found covering the surface it is skimmed off with a sponge or piece of blanket.

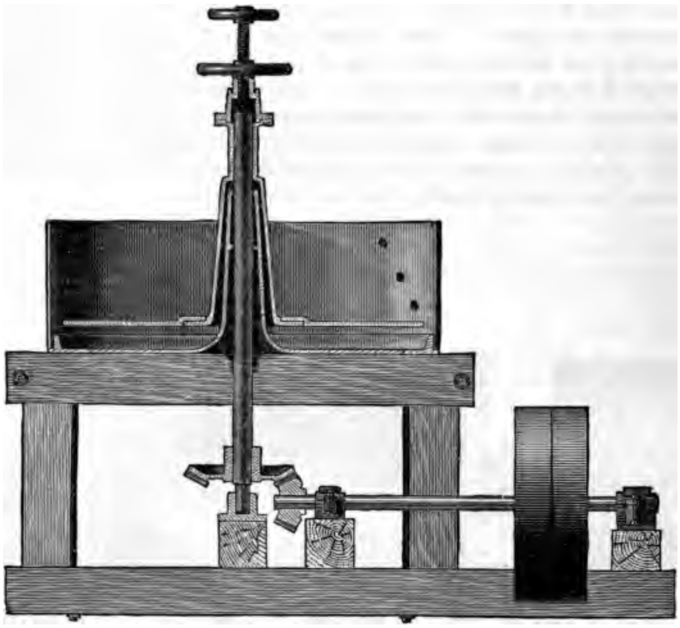

FIG. 27. CLEAN-UP PAN.

*Cleaning-up Room.*—This is an apartment in close proximity to the batteries and aprons, provided with a tight floor, and with a door under lock and key; the floor is best when laid in cement, to avoid all losses from split quicksilver or amalgam. It should be well lighted, and furnished with a sloping table large enough to place a screen frame on; also, with one or two water-tight boxes about 4′ long, 3′ wide, and 3′ deep, for panning-out purposes; these are supplied with plug-holes near the bottom, to drain off the water, besides water-pipes and fittings to fill the boxes when required. One or two wide shelves should be provided to hold the chemicals, quicksilver, and utensils needed in cleaning up. The latter consist of pans, wedgewood mortar, brushes, scoops, cups, knives, chisels, rubbers, scrapers, and a supply of closely woven drilling or light canvas; the latter is used to squeeze the superfluous quicksilver from the amalgam. A good pair of balances, with a set of accurate weights, capable of weighing the amalgam and the retorted bullion, should also be provided. The table should be made of a solid plank, or a slab of slate or marble, supplied with a raised edge, and grooved around to drain into a pan placed on a shelf attached below the lower end; some tables are covered with an amalgamated plate. It is sometimes convenient to have a small safe in the clean-up room, but it is always better to have the amalgam delivered to the office.

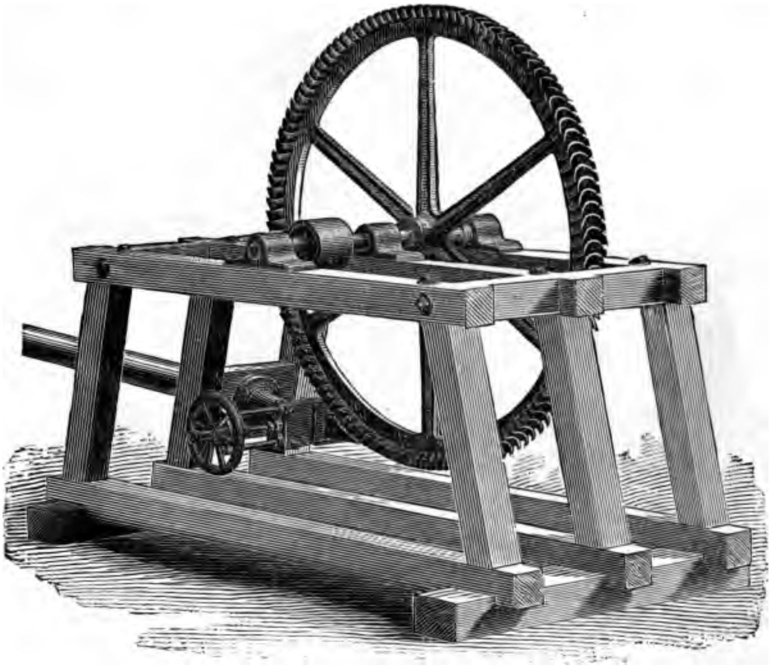

FIG. 28. THE KNIGHT WHEEL.

## POWER FOR MILLS.

On account of the favorable position of the majority of California mines as regards their proximity to mountain streams and the large ditch systems, the application of water for the motive power of the mills is rendered easy, and where the distance from these sources is remote, electricity generated in such localities and transmitted to the mill is being successfully applied. Where steam power has to be used, the well-timbered western slopes of the Sierra Nevada permit the cost of fuel to be kept at a comparatively low figure. Where both water and timber are hard to obtain, as in the desert regions of the southern part of the State, gas engines have been applied with most satisfactory results.

In applying water power, where the pressure is sufficient, hurdy-gurdy wheels are chiefly used; these are vertical wheels with narrow breasts, having buckets of various patterns radially attached to the outer circumference, the water being projected through one or more nozzles against the buckets at a low point of the wheel, allowing the water to pass from the buckets as soon as the blow has been delivered. The principal patterns in actual use are the Knight, Pelton, and Dodds; the actual effective power developed by the Pelton buckets is given at about 75% to 80%. Where sufficient pressure cannot be obtained, the Leffel turbine and the overshot wheel are in use. As the Pelton wheel seems to find the most frequent application in California, it may be convenient for millmen to have the following rule, applicable to these wheels: When the head of water is known in feet, multiply it by 0.0024147, and the product is the horse-power obtainable from one miner's inch of water.

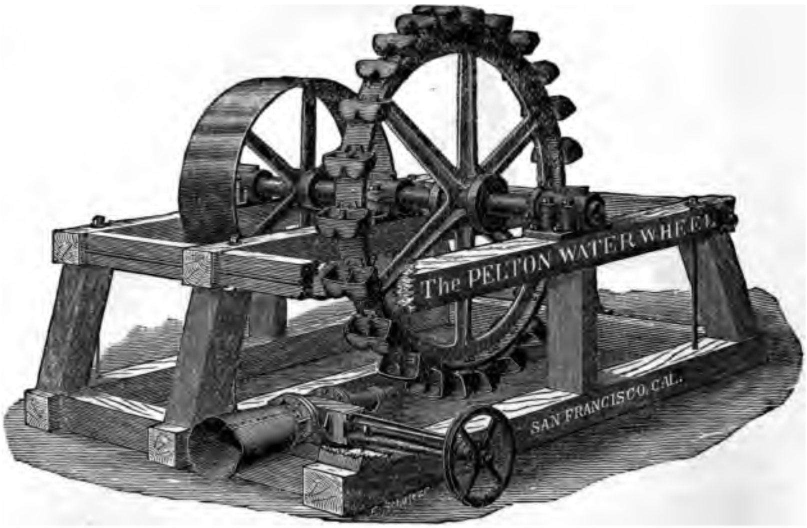


FIG. 29. THE PELTON WATER WHEEL.

The power necessary for different mill parts is:

| | |
|---|---|
| For each 850 lb. stamp, dropping 6″ 95 times per minute | 1.33 H.P. |
| For each 750 lb. stamp, dropping 6″ 95 times per minute | 1.18 H.P. |
| For each 650 lb. stamp, dropping 6″ 95 times per minute | 1.00 H.P. |
| For an 8″x10″ Blake pattern rock-breaker | 9.00 H.P. |
| For a Frue or Triumph vanner, with 220 revolutions per minute | 0.50 H.P. |
| For a 4′ clean-up pan, making 30 revolutions | 1.50 H.P. |
| For an amalgamating barrel, making 30 revolutions | 2.50 H.P. |
| For a mechanical batea, making 30 revolutions | 1.00 H.P. |

## MILL PRACTICES.

Where the conditions permit, it is becoming the custom to place the grizzly and the rock-breaker in close proximity to the hoist, so that the bucket or car on arriving at the surface is dumped direct on a grizzly, and the crushed ore is then run over the ore-bin in the mill, and emptied therein; where this is impracticable, the grizzly and rock-breaker are placed over the ore-bin in the mill.

The usual practice is to let the coarse ore from the grizzly drop on a platform on a level with the mouth of the rock-crusher, into which it is shoveled by hand; by this method the machine is not brought up to its full capacity. A better plan is to convey the coarse ore from the grizzly into the bin by means of a chute, having a sliding gate immediately above the receiving point of the crusher, and which is set so as to keep the space between the jaws always filled. In this way the work becomes automatic, and the services of the man attending the rock-breakers can be utilized in other parts of the mill during part of the time. Under such an arrangement the crusher will require more power, which should be independent from the other machinery. The rock-breaker is usually run during the daytime only, as it can crush in that time enough ore for the mill for the twenty-four hours.

The self-feeders, in a similar manner, are kept automatically filled

from the main ore-bin. The feeding through the tappet striking on the bumper-rod of the self-feeder, has of late been modified. A collar is fastened below the guides on the feed-stamp stem, taking the place of the tappet, thus avoiding the long bumper-rod. The gauging of the feed must be carefully attended to, if the stamps are to work up to their full capacity; there should never be more than about 1″ of rock between the stamp and die when they come together, or the feed should be just sufficient to keep iron from striking iron. When cleaning up the batteries, the self-feeders are drawn back on a track toward the ore-bin, giving access to the back of the mortars.

In preparing the mortar for ore-crushing, an inch or two of tailings is spread evenly over the bottom before putting the dies in place, as this saves the wear on the bottom plate. After the dies are placed exactly under each stamp, crushed ore and fine rock are banked around them to retain them in proper place until the sands have settled firmly about them. Care must be observed to keep the tops of *all* the dies at the same level at all times, as otherwise, when the stamps are dropping the highest die will strike against iron, while the others are still supplied with sufficient ore; this is known as "pounding."

The stamp-head, or boss, is now placed on the die with the small conical opening at the top, and the stem lowered into it, iron against iron if it is a close fit, and driven in solid. In case the connection is not tight, canvas strips about 2″ wide are laid crosswise over the opening before the stem is lowered. The stem, with the stamp-head, is now raised until the latch-finger catches under the lower face of the tappet and holds them suspended, and the shoe placed on the die. If the stamp-head hangs too low to permit of this, the stem is raised, and a block placed on top of the finger for the tappet to rest on. Narrow wooden wedges, about 1″ wide, the length of the neck of the shoe, and of the requisite thickness to fit tightly into the conical opening at the bottom of the stamp-head, are arranged in place and tied with a string. The block and finger are then removed, the stamp-head dropped over the shank, and wedges driven down firmly. This is done best by revolving the cam-shaft slowly, and, while placing the cam-stick between, permitting the cam to act on the tappet, raising and dropping the stamp until the lower edge of the stamp-head is nearly in contact with the shoulder of the stamp. It is not advisable to permit them to come solidly together, as it tends to loosen the iron ring that reinforces the stamp-head. A quick and convenient method of placing the wooden wedges on the shoe, is to cut a piece of canvas to fit exactly around the neck, and attach the wedges to the canvas by driving a tack through each one into the cloth. By keeping a supply of these on hand, it becomes an easy matter to encircle the shank on the shoe and tie them fast, should the shoe become loose and drop off while the mill is running.

*The Drop.*—The next operation is fixing the distance through which the stamp is to drop before striking the die. In most mills this distance is uniform for all the stamps; but, as previously stated, occasionally the stamp operating the feed, as also the two outside stamps, receive a greater drop.

The right height to give depends on the nature of the ore, as also on the speed to be given to the stamps; that is, the number of drops per minute. The tendency in most California mills is to run at a high rate

of speed, usually in the neighborhood of one hundred drops per minute. The height varies from 4″ to about 10″, generally but little, if any, above the water-level in the mortar.

In arranging the stamps for an equal drop, wooden blocks, cut about ½″ longer than the drop the stamps are to receive to permit the cams to clear the tappets, are placed on the die, between it and the shoe. Pieces of 2x4 scantling, cut to the desired length, answer well for the purpose. The keys in the tappet are loosened with a drift made of steel, the size of the keyholes, and used only for that purpose, and the stem is allowed to slip through the tappet until the shoe rests on the top of the wooden block beneath; or, if the shoe was resting on the block previously, the tappet is slipped up till resting on the latch-finger, when the keys are driven home solid. Care must be exercised not to drive the keys too solid, else there is danger of splitting the tappet. For the convenience of the millman, a chalk mark is made around the stem just above the tappet, which enables him, while running, to at once detect if any of the tappets have slipped. Should this occur it must be immediately re-set, or the battery work will be irregular. The battery-plates and chock-blocks are next put in place and keyed.

*The Discharge* is next arranged. This is the distance between the top of the new dies and the lower edge of the screen, and to fix the right distance is of importance. The greater the height of the discharge, the greater will be the proportionate amount of pulp and slime, and they also will be retained longer in the mortar. The quantity of amalgam retained in the mortar is also proportionately greater. A low discharge calls for a coarser screen, and naturally results in a larger output of the battery, and with a larger proportion of outside plate amalgam. With a *constant* height of the screen, the natural wear of the die increases the height of the discharge. For ordinary iron shoes and dies, and average rock, the wear of the die is roughly estimated from ¾ to 1 lb. of iron per ton of ore crushed. To counteract the effect of this wear on the discharge height, different-sized chock-blocks or screen-frames are supplied; the highest being used with new dies, and later replaced by lower ones, thus holding the distance more even than the use of a single size would permit. In some mills, when the dies are worn down, an iron plate, made for the purpose, is laid beneath them to raise them up.

As a very high discharge, besides creating much slime, beats up a larger portion of the gold into float gold than would be the case with low discharge, the choice necessarily influences the gold recovery; this is more particularly the case, if the ore carries any appreciable amount of valuable sulphurets. The discharge varies in the different mills from 4″ to 10″, the average being from 6″ to 7″.

*Screens.*—In fastening the screen to the screen-frame, care must be observed to get it on smooth, without any wrinkling or buckling. Tin screens must have the tin burned off before fastening to the frame; it is also well to expose the Russian-iron screens to a quick fire of shavings, to burn off the oil with which they are more or less faced. The edges of the screens are tacked to the frames, and are faced with strips of blanket to make a close connection with the mortar. In fastening a wire-cloth screen, to get it on smooth, a good method is to tack it first along the lower edge, then draw it up tight and even over the upper

edge, and nail it before cutting it off the roll. As previously stated, brass-wire screens should not be used in conjunction with cyanide of potassium, as the brass becomes coated and clogged with amalgam. The screen-frame with screen is dropped into the grooves cast on the outside of the mortar discharge, and fastened solid with iron wedges—two vertical (one for each groove) and a horizontal one in the center of the lower lip. The wedges should have a broad head, to facilitate knocking them out. After the screen has been fastened in place, a piece of canvas or a board should be hung in front to arrest the outward throw of the pulp from the drop of the stamp, and direct it in an even flow onto the plates beneath. In some mills this board is given a slope toward the screen, and has an amalgamated plate screwed on, which receives the splash. Bolted to the front of the modern mortars is a frame to carry the outside battery-plate and a distributing-box, a few inches above the apron-table on which it discharges.

When everything is ready to drop the stamps, the self-feeder is rolled to its place, the cam-shaft is set to revolving slowly, the water is turned into the battery, and the millman, standing on the platform above, grasps the hand-hold of the first finger or prop and introduces, with the other hand, the cam-stick between the tappet and the revolving cam; by this means the weight of the stamp is taken off the prop, which is pulled back and rested against the edge of the platform. This operation is repeated with each stamp until all are working. To carry out this operation when the shaft is revolving rapidly, without injuring the operator's hands, requires practice. The cam-stick mentioned above consists of a piece of wood about 2½′ long, 1″ thick at the point, running up to 2½″ near the handle, and faced with strap-iron or a strip of belting. It may also be made entirely of strips of belting, 2″ or 3″ wide, nailed over each other and attached to a wooden handle. To hang up the stamps, the hand-hold is grasped, the knee pressed to the latch-finger, and the cam-stick introduced between cam and tappet as before, and the latch-finger pushed under the tappet.

Before dropping the stems the face of the cams should be lightly lubricated, for which purpose axle grease or specially prepared compounds are used; a very useful one is a mixture of graphite and molasses; in some mills, to avoid the use of grease, the face of the cam is rubbed with a bar of common soap.

*Grease.*—It being essential, for good amalgamation, that the presence of grease be avoided in the battery, care must be observed in lubricating the cams, the stems where passing through the guides, and the shaft bearings. In many mills, trays made from old oil-cans are fastened beneath the bearings, cloth aprons are tacked from the underside of the guides to the floor above; rings of rubber packing or old belting also encircle the stems at the lower edges of the guides. The millman should diligently wipe off the stems and any part of the battery frame, where the presence of grease is indicated, at least once during a shift. Grease in the mortar is indicated by a black, dirty appearance of the surface of the plates, as also by the adhesion of more than the usual proportion of the amalgam to the iron castings inside the mortar. The usual remedy is to shut off a part of the battery water, for a short time, while adding a lye solution; or to add fine wood ashes to the ore.

*The Amount of Water Required* for the proper working of the battery depends on the nature of the ore; clayey ores, or such as have a high percentage of sulphurets, requiring the most; but while in the former case a greater amount is needed inside the mortar, the latter condition permits a part being added outside the screen, on the lip of the mortar. A small sluice-box, with plug-holes, is placed across the front in this case, or the water is conveyed by means of a half-inch perforated iron pipe, attached to the vertical supply pipe by an elbow joint, permitting it to be turned either way as required. "The amount of water used per ton of ore stamped varies from 1,000 to 2,400 gallons, with a mean amount of about 1,800 gallons per ton of rock crushed." * Most of the mills in actual practice figure roughly on one miner's inch of water, more or less, per twenty-four hours for each battery of five stamps. To obtain the largest amount of crushing of clean quartz from a battery, only sufficient water should be used inside to keep up the regular, even swash of the pulp, and if that be not sufficient to keep the plates on the outside clear from accumulating pulp, more may be added outside the the screen. The pulp, in passing down over the apron-plate, should roll in successive waves, corresponding to the back and forth wave-motion inside the battery, rather than flow in an even sheet, as affording a better opportunity of contact for the particles of amalgam.

Where the temperature falls low in winter, arrangements should be made to deliver the water in a tepid condition, as better amalgamating results will be obtained, through keeping the quicksilver in a lively condition. Where steam power is used, this can be easily arranged; but when using water power, a separate heater is required.

*Feeding.*—Hand-feeding has become nearly obsolete in California. It is only practiced in small concerns, or where a temporary mill has been put up for prospecting purposes. The advantages of a machine-fed mill are numerous; the chief of these are (1) that the wear of the iron of the shoes and dies is less and more even-faced; (2) that from 15% to 20% more ore can be crushed in a given time; and (3) that the labor expenses are reduced. The machines should be carefully gauged and watched to insure a steady, low feeding of the stamps. In order to insure a good splash in the mortar attention must be given to the succession in which the stamps are made to drop. A good splash is one that shows a wave passing along the lower edge of the screen, moving backward and forward from end to end, or a similar wave-motion that has its initial point from the center stamp. The succession most frequently adopted in California is 3, 5, 1, 4, 2; 1, 5, 2, 4, 3; 1, 3, 5, 2, 4, and 1, 4, 2, 5, 3; the last spreads the pulp very evenly from end to end. The greatest amount of discharge is obtained, apparently, by dropping the center stamp first; while the most crushing is done, other conditions being equal, by dropping the end ones first. Any arrangement of the stamps will answer, however, that distributes the pulp evenly and discharges it well.

*The Apron* should be set immediately in front of the mortar, but independent of the battery-frame, to exempt it from the jar of the stamps; it should be arranged to permit of the grade being easily altered if necessary. The size, shape, and grade of the apron-plates

* See VIIIth Report of State Mineralogist, p. 710.

differ widely, depending largely on the millman's preferences and experience. The usual form of the apron is rectangular, of the width of the discharge, and any length desired, but usually from 4′ to 12′, forming a level (transversely), smooth surface, set on a grade varying from ½″ to 2½″ to the foot. Sometimes the surface is divided by steps, with or without distributing-boxes. These are usually from 1″ to 2″. The apron should never be drawn in at the lower end, for reasons given farther on; and the steps should not be too deep, as otherwise the plate next to the drop will show mostly bare copper through scouring.

On examining a plate that is in use under good working conditions, it will appear that the upper portion, immediately below the mortar, say for a distance of 18″, carries at least 75% of all the amalgam caught on the apron, the largest accumulation showing along the line of impingement next the lip of the mortar. Now if the apron-plate were discontinued at about 2′, and continued again on a lower level of about 2″, a second line of accumulation would result, naturally on a smaller scale; hence, the advantage of the step form. Another advantage in this style of apron, is that by fastening these sections to the table by means of wooden buttons on the sides instead of cleats and screws, and having one extra plate on hand, the scraping and dressing of the same can be performed at any time, without stopping the crushing of the stamps, by removing the plate and substituting the extra one.

The grade of the apron-plate should be such as to keep the surface clear from any pulp accumulations, but not steep enough to obtain a scouring action. It will depend on the coarseness of the pulp, the nature of the gold, the amount of water available, and the percentage and nature of the sulphurets. Where a battery-plate is in use above the apron, it is usually given a grade of from 1½″ to 2″ to the foot. Grades for the apron proper vary from ½″ to 2½″ to the foot, but the average is about 1¼″. The apron-plates are usually silver-plated copper plates, which have largely superseded the copper amalgamated plate of former days—chiefly on account of the readiness with which the former plates do their full duty from the first starting, which is not the case with the copper plate; also, on account of their freedom from discoloration by oxidation. If silvered plates be used when running a very low-grade ore, the plating soon wears off, requiring a replating about every six months. The usual amount of silver put on plates is one ounce to the square foot. The usual thickness of copper plates is $\frac{1}{16}$″ to ⅛″. In preparing them for amalgamation they should first be carefully heated to a black heat, and plunged into cold water, which makes them soft and more ready to take up quicksilver. They are then scoured bright with fine tailings-sand, moistened with some cyanide of potassium, and applied with a block of wood; then dressed all over with a weak solution of nitric acid, or with cyanide of potassium and quicksilver, with sodium amalgam sprinkled over and brushed or rubbed into the surface. Before final use, it is well to give them a coating of fine gold amalgam; or, if not convenient, silver amalgam will answer. In using the cyanide of potassium solution, care must be taken not to use it too strong, especially if the quicksilver is not applied to the plate immediately, otherwise a coating is formed on the surface that will not take up the quicksilver. Where the ore is of a fair grade, after a long period of continuous use the plate will have absorbed an amount of gold

that will not yield to scraping unless the plate is immersed in boiling water for a time before being scraped, or heated over a fire and hammered with a mallet on the reverse side, in which case care must be taken not to dent the plate.

As the saving of amalgam on the apron and sluice-plates is largely a matter of gravity, the conditions under which the pulp passes over the plates should conform to the laws pertaining to the falling of a body through a moving liquid medium; hence the proper shape of the apron, and the flow and consistency of the pulp, should be well considered. If, as was formerly the almost universal custom, the lower end of the apron be contracted (and in numerous cases this contraction was as great as four to one), the depth of the pulp spread over the surface of the plate increases as it passes down; the flow of the water across a given section becomes uneven, forming at the sides a swirl, along the edge of which, sand is precipitated, covering and rendering that portion of the plate useless, from its inability to come in contact with the particles of amalgam, while producing scouring currents at other parts. The proper method is to spread the flow over a wider surface as it passes from one plate to the other, and lessen the grade, which may require an addition of clear water.

This contraction of the plates is made to this day in most of the mills, when connecting with the sluice-plates. The liquid pulp, starting with a width equal to that of the mortar-discharge, is made to pass over sluice-plates from 1′ to 2½′ in width; hence, the comparatively small percentage of amalgam obtained from them. The only condition under which narrower plates are permissible, is where, previous to receiving the pulp, a certain amount of the solid matter has been diverted. Where all the pulp goes from the plates to concentrators, the latter become an important factor in regulating the amount of water turned into the battery. The feed-water required for concentrators of the vanner types is from one to two gallons per minute.

In dressing the apron-plates prior to starting the stamps, they are first washed down with the hose, to remove all particles of coarse sand which might otherwise scratch the plate during the subsequent dressing, then rubbed with a brush, using, if necessary, some fine tailings-sand to remove all spots or stains. During this part of the operation, the brush is moistened with different chemicals, according to the preference of the millman; some use weak cyanide of potassium; others use strong brine, with a small addition of sulphuric acid; also, sal ammoniac, or soda, or lye, besides other combinations. In many cases these prescriptions are carefully guarded by their possessors as trade secrets, and are considered the basis of all the success the owner has achieved in his business. Anything that will give the plate a clean surface, free from oxidation stains, and retain for the quicksilver its bright condition, is useful in this respect. The main point to achieve success is to always keep the amalgam on the plate bright, and of the right consistency, and this art can only be perfectly acquired by actual practice around the battery and plates. After the plate has been thoroughly cleaned, quicksilver is thinly sprinkled over the entire surface, through a cloth, and spread evenly by means of a brush or piece of blanket, and finally the surface gone over with a soft broom or brush, from side to side; this leaves the amalgam remaining on the plate with fine ridges parallel to the screen.

Among the plate devices used in California mills, which may take the place of the apron-plates, or may follow them, is a late invention known as the *Gold King Amalgamator*. It consists of an iron cylinder, or drum, 6′ long and 12½″ in diameter, divided lengthwise into two equal parts, hinged together, and capable of being locked. Fitting tight inside of the cylinder are two corresponding semi-cylindrical silver plates, each with two longitudinal ribs set radially, at one-third distance apart and about 3″ deep. The upper end of the cylinder is furnished, around the


FIG. 30. GOLD KING AMALGAMATOR.

circumference, with tooth-gearing, into which fits a spur wheel with a four to one transmission, driven by a 12″ pulley. In the center of this end is a 3″ feed opening, through which the pulp is dropped into the revolving cylinder. A trunnion at the lower end rests in a slide bearing, that permits of fixing the grade to be given the cylinder by means of set screws. The machine makes forty revolutions per minute, the pulp requiring about 3½ minutes to pass through the machine before being discharged. It is run by less than ¼ H.P., and is easily set up.

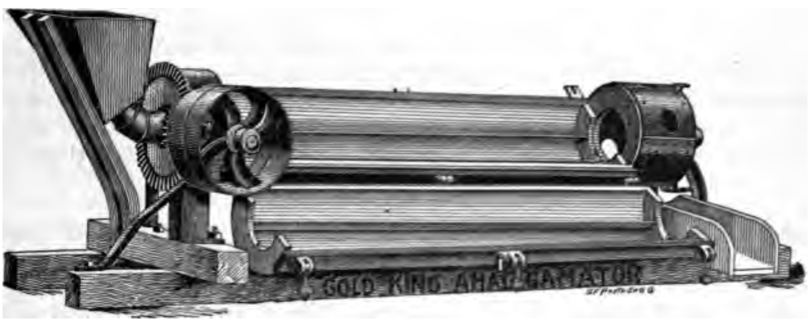

FIG. 31. GOLD KING AMALGAMATOR.

The pulp, when dropped in the closed cylinder, is caught by one of the ribs and raised to the highest point, when it drops, to be again taken up by the next rib, advancing at the same time a short distance ahead. The discharge is through the center at the lower end of the cylinder. From 15 to 20 tons can be passed through in a day; or for a larger sized machine, from 25 to 40 tons.

Where concentrators are used in the mill, the sluice-plates that follow the aprons are usually not over 8′ in length and from 16″ to 20″ wide,

with less grade than the apron. This latter point is reversed in some mills, and the sluice-plates are comparatively steep.

Between the aprons and the sluice-boxes a drop box is placed, into which the pulp from the aprons discharges; there is one to each apron, or one for two adjoining. ones. These boxes are 1′ wide and about 10″ deep, with flat or partly sloping bottoms; these latter, generally where one box is used for two aprons, the bottoms sloping from each end across the width of the apron, toward a central part where the bottom is level, and from whence it passes by overflow to the sluice-plates. These sluice-plates are in short lengths, and are either laid overlapping or screwed down to form a continuous sheet, and are prepared and treated in the same manner as the aprons. Of late years a useful

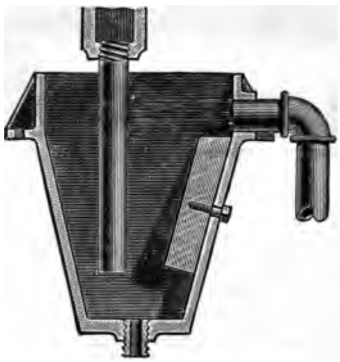

FIG. 32. AMALGAM TRAP.

addition is being made to the plates in the form of a shaking-plate, of the same width as the aprons, and immediately below them. It is either suspended or on a movable frame, and is given an end or side-shaking motion and light grade; for an end shake, the motion is imparted by a cam with ⅝″ stroke, and two hundred strokes per minute. The correct strokes for these plates must be determined at each mill by experiment. Their efficiency was demonstrated in one mill, where the pulp passed over two consecutive apron-plates, and then to the shaking plate, which accumulated a greater amount of amalgam then the second apron.

*Amalgam Traps* (Fig. 32).—To retain any quicksilver or small particles of amalgam that escape inadvertently while dressing or cleaning the plates, traps are generally placed below the sluice-plates, and are made of various patterns. The general idea is for the pulp to drop to the bottom of a deep vessel and flow out at or near the upper edge; in some cases, passing over a series of inclined shelves of copper plates during the descent. A simple and very efficient contrivance for an amalgam trap is to suspend a narrow box by one end and attach the opposite end to a rod connected by a pin to an eccentric, through which it receives a gentle shaking motion in the direction of its long side. The tailings are introduced into a stationary box immediately above, from whence, diluted with fresh water, the pulp passes over the top of a partition in an even sheet to the suspended box below. The proper motion for this lower box must be found by experimenting, for which purpose

the end of the rod is supplied with a series of holes, to shorten or lengthen the stroke. The motion should be just sufficient to keep the pulp suspended like quicksand, without splashing or caking on the bottom.

*Amalgamating.*—Quicksilver is charged by hand into the mortars through the throat, at stated intervals, with a small wooden spoon. Automatic quicksilver-feeders have been invented that are worked from the cam-shaft in such a manner that, at stated intervals, a little cup on a ratchet wheel, in revolving, dips quicksilver from a reservoir and drops it through a tube into the mortar. This insures absolute regularity; but for some reason they do not find much application in California. Retorted or new quicksilver should be used for charging as well as for dressing the plates. It is a good plan to keep the quicksilver used for these purposes covered with a weak solution of cyanide of potassium.

*Quantity of Quicksilver.*—To form some idea of the amount of mercury necessary to be introduced when handling an ore, the value of which is not known, a horn-spoon test of a weighed quantity is made, and the quantity of gold decided. Gold alloyed with an appreciable amount of silver requires a larger addition of quicksilver than does a purer gold. One ounce of gold of average fineness can be amalgamated with 1 oz. of quicksilver, but for a safety margin, an allowance must be made, so that 2 ozs. will answer better; and with extremely finely divided gold, 2½ or 3 ozs. If the stamps have a duty of two tons each, the amount of mercury requisite to amalgamate the gold contained in one ton of ore should be divided into five parts and introduced at half-hour intervals. If the ore be of low grade, the necessary portion may be added every hour; as the value increases, the stated intervals for charging should be reduced. The larger proportion of California gold ores receive mercury every half hour.

The skilled millman judges from the condition of his plates as to whether he is charging correctly. He places his finger on the apron-plate, and if the accumulated amalgam gives to a gentle resistance, and has a putty-like feeling, the condition is about right; when hard to move, he must increase the charge; or if thin, reduce it. The harder the amalgam, the more it assumes a dead-white color. The matter of correct charging of the mercury requires a constant watching, as on this depends the success of battery amalgamation; hence, the ore should be frequently tested with the horn-spoon.

Amalgam retained on the inside battery-plates weighs heavier, for the bulk, than the apron amalgam. There is a diversity of opinion among millmen as to how often the amalgam accumulated on the aprons and sluices should be removed. Thus it is found in the California milling practices that aprons are scraped as often as twice a day in some mills, while in others it is allowed to accumulate from one clean-up day to the next, which sometimes means once a month. Personal experiments by the writer, conducted in various mills, invariably showed a yield of more amalgam from the more frequent removal of the accumulations, but as the clean-up of the apron would require the cessation of crushing, such frequent stoppages would materially lessen the output. To avoid this, as the upper 18″ of the apron-plate retains about 75% of all the

amalgam on it, this much of the apron-plate may be made separate from the rest, and held in place by wooden buttons on the side, so that it can be removed at any time while the battery is at work, and an extra plate, provided for the purpose, slipped in its place. Once or twice in the twenty-four hours it is advisable to hang up the stamps, one battery at a time, and dress-over the surface of the apron-plate, sprinkling, if necessary, a little fresh mercury, and brushing it into the adhering amalgam, after which the amalgam should be evenly spread out again. This takes but very few minutes. Frequently, when dressing a plate, a very fine coating of a brownish or grayish color can be seen adhering to the surface, which, on the application of the brush, is easily detached and thoughtlessly washed off. If this be examined under the glass, it will be found to contain considerable gold, hence should be gathered carefully in the gold-pan.

To remove the amalgam from the plates, the stamps are hung up, the battery-water shut off, and the front of the screen and plates hosed off to remove any sand which would scratch the plate. The surface of the plate is softened by the addition of quicksilver until the amalgam moves readily. Then, commencing at the bottom and working upward, with a piece of rubber, or rubber belting, 4″ long with square edges, the amalgam is pushed ahead to the upper end of the apron, gathered in a heap, and transferred to a pan or bowl by means of a scoop. The amalgam is taken to the clean-up room for further cleansing.

Where the amalgam has been retained on the plate for any length of time, as during an entire run, it requires a chisel or case-knife to remove it thoroughly, care being taken not to scratch the plate. In scraping a plate it is not advisable to remove ("skin") all the amalgam; enough should be left to form a thin coating, when ready to commence crushing again.

All mills experience more or less loss of quicksilver, partly through careless handling in dressing the plates, but also from the "flouring" of the mercury (breaking up into minute globules) after charging in the battery. This loss is extremely variable in the different mills, depending on the nature of the ore, high discharge, and low temperature of the battery-water. Ores carrying much talc, black oxide of manganese, galena, or arsenical pyrites, cause a good deal of flouring of the mercury. A further cause of loss is through incomplete retorting, a certain amount of mercury being retained in the bullion, which is volatilized in the subsequent melting. One half ounce to the ton of ore may be taken as near the average loss for California mills, although in a few cases these figures are doubled.

As the amalgam retained in the battery is less liable to loss than that portion adhering to the outside plates, the aim of the millman is to retain the largest proportion inside the screens. The coarseness of the gold has a good deal to do in this direction, as well as the splash and height of discharge. In some mills, as high as 80% of the total yield of amalgam will be found in the battery; it is always greatest, with the same grade of gold, where the most copper-plate surface is found inside the battery. The average proportions of amalgam retained in this country may be stated as two thirds in the battery as against one third on the (outside) plates, depending, of course, on the character of the gold in each district.

As the proper condition of the mercury is a matter of importance to

the millman, it is well to become familiar with its different phases. Pure mercury is bright, quick, and does not change its appearance on exposure to the air at ordinary temperature, but evaporates slightly. As the temperature decreases it becomes stiffer, and at low temperature assumes a more leaden appearance; in raising the temperature it becomes more liquid. At 60° Fahr. it emits vapor sufficient to discolor a bright piece of gold when suspended over it in a closed vessel. Pure mercury, if dropped into a porcelain dish or on a table, will form into spherical globules, whereas the impure metal breaks into pear-shaped drops, and if very impure, the particles drag a tail when moved. If containing lead, a skin of metal will remain on the fingernails when passing the hand through the surface. The introduction of grease or unctuous substances, like clay and talc, incline the metal to separate into extremely fine globules—flouring. Quicksilver is attacked by heated concentrated sulphuric acid, but is not affected by it when diluted. Muriatic acid likewise does not affect it. Nitric acid attacks it and forms nitrate of mercury, a white compound. Quicksilver that has been used in gold-milling dissolves and retains a certain amount of gold, which remains with it, even after retorting. If quicksilver of this description be left for months undisturbed in a cold place and then carefully poured or siphoned off, a network of fine, needle-shaped crystals of amalgam will be found in the bottom of the vessel, derived from this gold held in solution.

*Sodium Amalgam.*—As sodium amalgam is frequently added to the quicksilver by millmen, the following method of preparing it is given: Dissolve small, dry chips of clean sodium, freshly cut from a stick, in pure, dry mercury, gently heated in a flask or porcelain dish; add it piece by piece until the mass has attained the consistency of soft putty, which should always be kept dry and well bottled, as it deteriorates rapidly in the air. This preparation is added to the mercury when dressing the plates; and to know when the proper amount has been added, dip a brightened nail into the quicksilver, which will adhere slightly to the edges of the nail if the amount be correct; whereas, if it becomes entirely coated, too much has been used, and more quicksilver must be added; on the other hand, if there be no signs of adhesion, more sodium amalgam must be added.

Nearly all commercial mercury needs cleaning. The handiest way is to digest with dilute nitric acid for twenty-four hours, taking one part of acid to three of water. In retorting foul quicksilver to purify it, the retort should only be half filled and the quicksilver covered with a layer of quicklime or charcoal powder. The heating should then be done very gradually, the retort not being brought to a full red heat.

*Cleaning Up.*—When ready to clean up a mortar, the feed of ore is shut off, and the speed of the stamps reduced until as much of the sand, etc., as possible has been discharged and iron strikes on iron. The battery-water is then shut off, the self-feeder pushed back, the stamps hung up, the splash-board or canvas removed from in front of the screen, and the face of the latter washed off with the hose. The aprons and plates are then scraped, and the aprons, if fixed, covered with planks near the mortar, to protect them while working around the mortar. The keys that hold the screen in place are withdrawn and the

screen-frame loosened and slightly raised, permitting the water that is still retained in the mortar to gradually run out; a too sudden raising of the screen-frame from the chock-block would cause the water to escape in a body and possibly wash amalgam from the plates. After raising the screen out of the grooves, the chock-block and inside plates are removed and all of them carefully washed over the apron, scraped, and set to one side or removed to the clean-up room for treatment. The sand mixed with ore on and around the dies is taken out by trowels and passed through some other mortar, or retained to place around the dies when returned to the mortar. The dies are broken out of their beds with the help of chisels and crowbars; when the center or end die has been successfully worked loose, the remaining ones are easily taken out, washed, examined for any adhering amalgam (which is scraped off), and placed on the floor, *in the same order* they occupied in the battery, ready to be replaced. The remainder of the material in the mortar is then easily removed, and placed in the clean-up barrel; in small mills it is panned in a water-box provided for the purpose in the clean-up room. In the revolving clean-up barrel, pieces of quartz or old iron, with an additional amount of quicksilver, are added, and the barrel is half filled with water, where it is left revolving for a couple of hours. As all battery sands contain more or less nails and chips of iron and steel, these are removed by a magnet while panning out. The clean-up barrel is discharged through a manhole into a bucket placed over a riffled sluice. The bulk of the quicksilver and amalgam is retained in the bucket, and the overflow passes into the sluice.

After all the sand, etc., has been removed from the battery, the inside is washed out, and any amalgam found adhering to the sides or linings is carefully scraped off with a case-knife and placed with the rest of the amalgam for further cleaning. A bed of dry tailings-sand is then spread over the bottom of the mortar, and the dies *replaced exactly* as they were before. The tappets are then set, plates and screens put in, the feeder replaced, water turned on, and the battery once more started.

The operation of cleaning-up the batteries is performed usually once or twice a month, and in some mills once a week, at which time tappets are re-set and any necessary repairs made; also, any shoes that are too thin or broken are knocked from the boss and new ones substituted. As one new shoe in a battery of old ones causes irregular working, it is best to replace all the shoes at the same time, and if any of them are not worn down thin enough to discard, they may be set aside and used to replace a broken one at some future time. The same thing holds good with the dies, for if they are of uneven height they interfere with the regularity of "splash," and the higher die will be pounding iron while the remainder have still a sufficient cushion of quartz.

The amalgam obtained from a clean-up is washed in small batches in the gold-pan, to free it from all sand, fine iron, or sulphurets, and then stirred up with an excess of mercury in a wedgewood mortar, bringing all impurities to the surface; this dross is skimmed off and collected for further cleaning. The superfluous quicksilver is squeezed through a straining cloth or closely woven drilling, or through buckskin, and the resulting balls of amalgam retorted. This squeezing is best done by hand. After first thoroughly wetting the cloth or skin, it is laid loosely over a cup or bowl, and a convenient amount of amalgam poured in the center, enough to make, when squeezed, a ball of 20

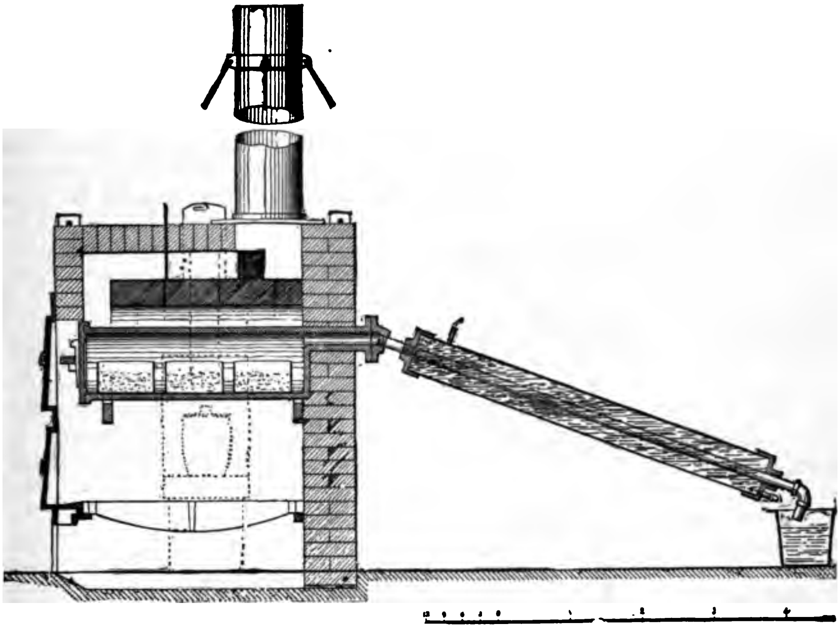

FIG. 33.

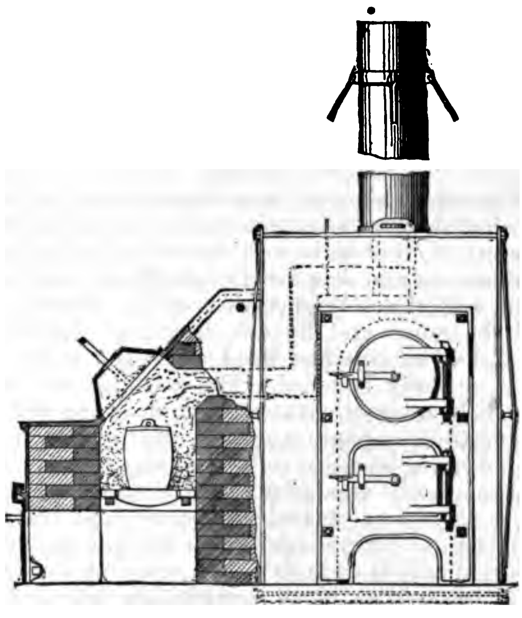

FIG. 34.

to 30 ozs. The ends of the cloth are then gathered tightly together, and commencing near the ends, it is twisted until the amalgam is compressed to a hard ball, the strained quicksilver dropping into a pan of water beneath. It is not good practice to squeeze the balls too dry, as the last quicksilver expressed is heavily saturated with gold.

In large mills the retorting is done in pans placed in an iron cylindrical retort built into a furnace, where the flame passes under and around it. (See Figs. 33 and 34.) But in the majority of cases in California they use the cup-shaped iron retort. (See Fig. 35.) These

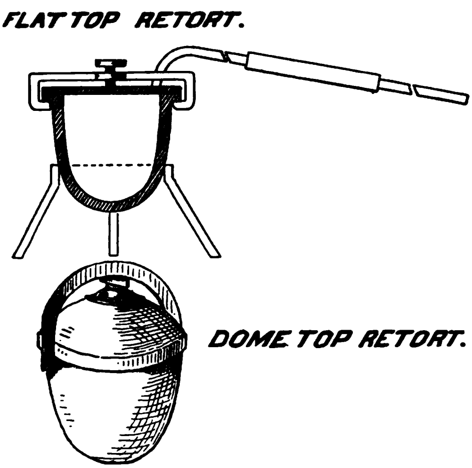


FIG. 35.

are made in different sizes, numbered from 1 to 7; No. 1 containing 150 ozs., and No. 7, 2,000 ozs. They are made of cast-iron, with flat or half-spherical lids, which are secured to the retort by clamps and wedges or thumb-screws, the flanges being ground together. From a vent-hole in the cover a curved condensing-pipe, securely screwed in, extends several feet. The retort is placed in a ring-standard, or suspended when retorting, and should always have a space of about 6″ beneath it. In preparing to retort, the inside is well rubbed with chalk and the balls of amalgam broken up and dropped in loosely; not pressed down into a solid cake, as is sometimes the practice, as that retards the operation. The flanges of the retort and lid are then luted together with a thin paste of flour and water or sifted wood ashes and water (the former is preferable), and securely fastened. The extended end of the condensing-pipe is placed in a vessel with water, and as this pipe must be kept cool, fresh water is kept passing over it during the entire operation. The retort should never be filled to its full capacity, to avoid danger of an explosion through the amalgám swelling and closing the vent. At first a light fire should be started at the top, and the heat gradually increased until drops of quicksilver issue from the end of the condensing-pipe. The retort should then be kept at a red heat until no more quicksilver is seen to issue from the pipe, when the temperature should be raised to a bright "cherry heat" for a short time. The retort should be kept covered by the fire during the whole operation. If during the

retorting the condensing-pipe should suck water, it should be raised momentarily out of the water to permit of the latter flowing out. A better arrangement, and one that obviates this difficulty, is to attach firmly to the end of the pipe, a rubber or canvas bag in the water, which will distend itself as soon as the mercury commences to flow, and collapse when the distillation ceases. When the operation is completed, which usually occupies about two hours, if the amount be not very large, the quicksilver is removed and the retort taken from the fire and allowed to cool; the lid is removed and the retort turned over a *dry* gold-pan. If the gold adheres to the retort, a few taps with the hammer on the bottom or the help of a long-handled chisel will release it. Well-cleaned and retorted amalgam should show a good yellow color. If black spots be seen it is proof that the cleaning was not thoroughly done, and a pale-whitish color shows that it still contains quicksilver. Care should be observed, when removing the lid of the retort, to avoid inhaling any fumes retained therein. All retorted amalgam should be melted and run into a bar, before shipping, as it saves losses incurred by abrasion where the distance is great to the shipping point. The melting is performed in a black-lead crucible, which, when new, must first be dried and annealed by placing the inverted crucible and lid in the furnace with a slow fire, which is gradually increased until the crucible is red hot. When ready to commence melting, the crucible is placed on a firebrick in the furnace, after introducing the retorted bullion, in not too large pieces, with borax, and covered with the lid, adding, if necessary, more of the bullion as the metal subsides. After all is melted down, the slag is skimmed off carefully from the top of the metal, which should show a bright surface. It is then ready for pouring. Should the surface not appear bright, but show a scum on top, some lumps of borax must be added, the crucible again covered and heated, when the scum will be slagged and skimmed as before, when it is ready to be poured into a mould. Should the second addition of borax fail to produce a bright surface, a very little niter may be added with the borax. Before using the mould it should be warmed and smoked on the inside by holding over the flame of a lamp or over a dish with burning rosin. The metal in the pot should be stirred before pouring; the stirrer, an iron rod, must be heated before introducing it. The bar, when solid, is turned out of the mould, and any adhering slag is hammered off; it can then be dipped into water to thoroughly cool it, dried, and weighed. Two small chips should then be taken—one from an upper corner, the other from the diagonally opposite lower corner—to be assayed.

### ASSAYING AND SAMPLING.

Although at present most California mills have their own assayers to test the ore and the tailings, the time was not so very remote when it was not considered requisite to do *any* assaying. The expert millman could tell (?) by horn-spoon test how much his ore would mill to the ton; and if a horn-spoon test of the tailings showed no amalgam, he confidently asserted that all was being saved. It was decidedly a case where "ignorance was bliss." No gold milling can be carried on understandingly without light being thrown on the different results achieved, and which can only be given by careful sampling and assaying. It is not sufficient to know that a certain loss has been sustained. It should

be accompanied by a knowledge in what particular part of the operation the loss has been incurred, to enable the operator to remedy the evil; hence, the necessity of constant sampling and assaying. In some cases the loss will be found entirely in the coarse sands, indicating that the screens are not fine enough; again, the loss may be entirely through sliming of the ore, or the missing percentages of gold will be found mostly in the sulphurets. The assay test alone, *with correct sampling*, furnishes the knowledge.

*Sampling.*—Samples should be taken regularly of the ore as it comes to the mill, as well as of the tailings *as they pass off*, for without the knowledge derived from these two operations there is no means of controlling the work.

Ore, as it arrives at the mill, is sampled by taking a stated amount (shovelful) from each ore-car or wagon, and throwing the samples together in a pile on a clean-swept floor or into a small bin. The pile should be shoveled over after breaking the pieces to the size of macadam; or if the pile be too large, cut through it at right angles, throwing the rock from the trench thus made in a pile by itself. This should be crushed or broken to a nearly uniform size, mixed by shoveling, and made into a low, truncated cone, which is divided into four equal parts by making a cross on the surface, and throwing out two diagonal quarters, which are again reduced in size, made into a second similar cone, and treated as before. This quartering and crushing is continued until a half-pound sample is obtained for fire assay. Great care must be observed when removing the different quarters to see that all the fine dust is swept up and added to the pile each time, as otherwise very defective results will be obtained. The rest of the ore is returned to the main ore-bin. Samples taken in this way from the aprons of the self-feeders are likely to give a more correct average, having been crushed, and the coarse and fine duly mixed. Samples should be taken at regular intervals from the pulp *with the water* that has passed over all the plates, and also from the concentrators.

Tailings samples should be taken at stated intervals by passing a vessel across the entire width of the discharge, where they leave the mill, without permitting any to flow over, and gathering at each interval an even amount. This is allowed to settle in a bucket and the clear water then poured off carefully or drawn off with a siphon. The residue is dried and thoroughly mixed, and several packages of 5,000 to 10,000 grains each weighed out. In some mills tailings samples are obtained automatically, using their current as the motive power for the sampler, which works by intermittently deflecting a spout through the tailings where they finally drop from the sluices, obtaining the sample across the entire section of the current. An appliance for this purpose is shown in the accompanying drawing. (Fig. 36.)

To ascertain the amount of slimes in the tailings sample, put one of the packages into a bucket, add water, and stir it. After settling two or three minutes, pour off the muddy water into a separate vessel; repeat this operation until the water comes off clear; add a little powdered alum in the vessel containing the muddy water, and when the mud has all settled, draw off the top water carefully and evaporate the remainder. Dry the washed sands of the sample, and pass through different sized

creens, weighing the different amounts as they have passed, and assay ach size; this will show where the greatest loss is sustained.

To ascertain where the loss in sulphurets occurs, it is sufficient to pass ne of the 10,000 grains samples through a 60-mesh wire screen; weigh hat which passes through and that which remains on the screen, and

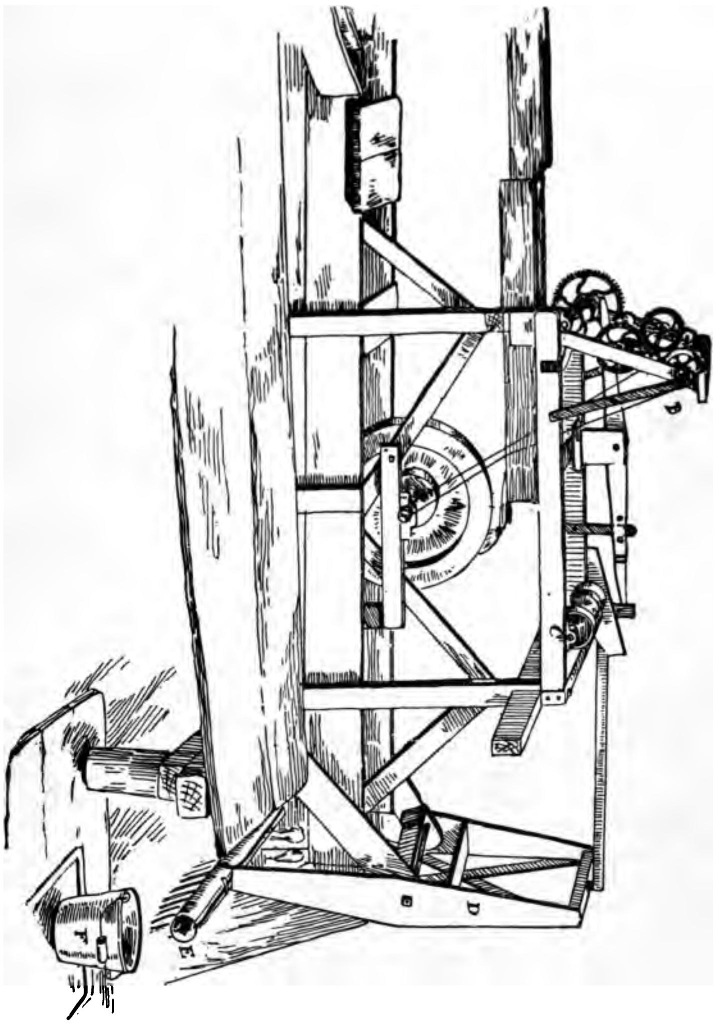

FIG. 36. EMPIRE AUTOMATIC TAILINGS SAMPLER.
Overshot water wheel actuates wheels B, which at stated intervals operates frame D, the tailings sample passing through E, into the bucket or receptacle F.

an out each lot carefully by itself, from one pan into another, as long s sulphurets can be recovered; then weigh each batch of sulphurets eparately.

The use of 10,000 grains is recommended, as every 100 grains is 1%, nd each grain is $\frac{1}{100}$ of 1%; it is also a convenient size for obtaining ccurate results. By using pulp samples instead of tailings, the amount f sulphurets in the ore may be ascertained.

If the sulphurets assay $75 per ton, and the quantity per ton is 1.7%,

the value of the sulphurets in one ton of ore is found by multiplying \$75 by 0.017, which would be \$1 27 per ton.. If the loss of sulphurets in the tailings is 11 grains out of the 10,000 grains sample, and the value of the sulphurets is \$75 per ton, then multiply \$75 by 0.0011, and the value of sulphurets in the tailings is found to be \$0.0825 (8¼ cents) per ton of tailings.

MILL ASSAYS.

*Amalgamation* (*Free-Gold*) *Assay.*—Take two pounds (being exactly one thousandth part of a ton) of ore, crush in an iron mortar, and pass through a No. 60 sieve; remove the gold and other metallic substances left on the sieve, and place in a small porcelain dish containing a little dilute nitric acid, to remove any adhering crusts of oxide of iron, etc., which might prevent amalgamation; these residues are then carefully washed and thrown into the sifted ore, which is then placed in a wedgewood-ware mortar and mixed with enough warm water to make a stiff paste. To an ounce Troy (480 grains) of new, clean mercury, *free from gold*, add a piece of clean sodium about the size of a pea. The mercury thus highly charged with sodium is then thrown into the mortar containing the sample, and the mass ground constantly for an hour, when amalgamation should be quite complete. The mass is then transferred to a gold-pan and *carefully* washed over another pan or tub, in which the tailings are caught, and re-washed to save anything which may have escaped. The mercury is collected and transferred to a small dish; if it be mueh floured and refuse to run into globules, stir it with a small piece of sodium held in the end of a glass tube, which will cause it to run together. The mercury is then washed carefully in clear water and dried with blotting paper. It is then re-weighed, and if the loss exceeds 5% the assay must be rejected and a new one made. The mercury is next transferred to a small annealing-cup or crucible, which has been carefully black-leaded inside, covered with a porcelain or clay cover, and volatilized with a gentle heat. When all the mercury has been volatilized, about 50 grains of assay lead are thrown into the crucible and melted, giving it a rotary motion while in a molten state. It is then removed, cupelled, and the "button" weighed. It may be assumed without sensible error that the mercury lost in the operation carried the same proportion of gold as is contained in the mercury recovered; hence the gold contents of the ore will be found by multiplying the weight of the "button" obtained by the weight of the original quantity of mercury, and dividing the product by the difference between the weight of the mercury recovered and the "button." This figure, multiplied by 1,000, gives the weight, in grains, of the free gold and silver per ton of ore, which, for all practical purposes, may be assumed to be all gold. Should, however, greater accuracy be desired, hammer the "button" flat and thin, and dissolve the silver from it with nitric acid, and weigh the gold. The difference in weight represents the silver.

*Panning Assay.*—Take 2 lbs. of ore, crush, and pass through a No. 40 sieve; any gold in the residue left on the sieve being set aside. The sample is then carefully panned, and the tailings re-panned, to make sure nothing is lost. This operation will show at once whether the ore is rich in sulphurets or not, and the nature of them. The visible gold should be panned as free as possible from all the sulphurets, taking care

that none is lost. The pan and its contents, together with the residue left on the sieve, are dried by holding over a fire; the contents are brushed into a cone of lead-foil, rolled up, melted, and cupelled. The "button" is weighed, and the free gold determined by multiplying its weight by 1,000.

The tailings produced in the panning operations should be panned over several times to collect all the sulphurets, which should then be dried, weighed, and their percentage in the ore determined.

Another method consists in not separating the free gold from the sulphurets, but in treating them both together by fire-assay, and determin-

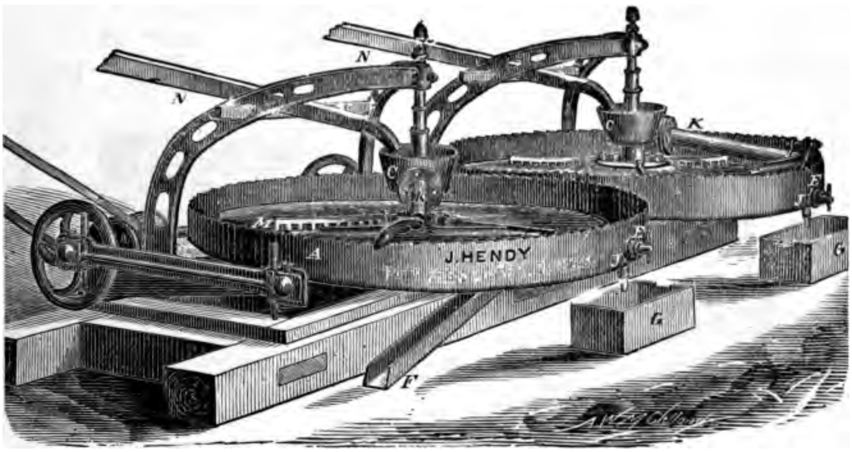


FIG. 36*a*. THE HENDY CONCENTRATOR.

ing the *total* value of the gold present in them. The operations, as far as described, are all that can be properly considered as coming under the term of battery amalgamation as practiced in California, if we except the use of the riffle and blanket sluices; these are placed below all the plates, and receive a very spasmodic attention in the majority of mills. Blankets are laid in strips, about 16" wide and about 6' long, overlapping each other in double sets of sluices, set on a grade of about ¾" to the foot, washed in a separate water-box. The material thus obtained, with the contents of the riffles, is deprived of its valuable contents by the aid of arrastras, pans, or Chili mills. But few blanket-sluices are found to-day in California mills.

On the practical development of the Plattner chlorination process, by Mr. Deetken, in the "sixties," it was demonstrated that many of the low-grade quartz veins carried enough gold in their sulphurets to make their working profitable, causing attention to be directed to the concentration of these ores by mechanical contrivances. From the constant and successful use of the gold-pan the mechanical application of a similar motion was sought, resulting in the use of the Hendy and similar concentrating machines.

The Hendy concentrator consists briefly of a shallow iron pan with an annular groove on the outer edge and a waste discharge in the center. It is supported on a central upright shaft passing through the center of the pan, on which revolves, above the pan, a central bowl to receive the

pulp, having two tubular arms extending close to the outer edge of the pan; these uniformly discharge the pulp at right angles from their axis. At a point on its circumference the pan is attached to a crank-shaft, making about 220 revolutions per minute. The sulphurets and small balls of amalgam gather in the groove at the outer edge, from whence they are drawn through a gate, which is regulated to be automatic in its discharge. This gate is not opened until the groove is pretty well filled with sulphurets. Two of these machines, driven by one shaft, are required for a five-stamp battery. The machine needs constant attention; one man can attend to twelve machines on a shift. They have been mostly displaced by the endless-belt machines which have developed from the endless blanket and shaking-table.

In 1867 the first patents for the revolving belt were issued.* This was the commencement of the belt concentrators, of which at present

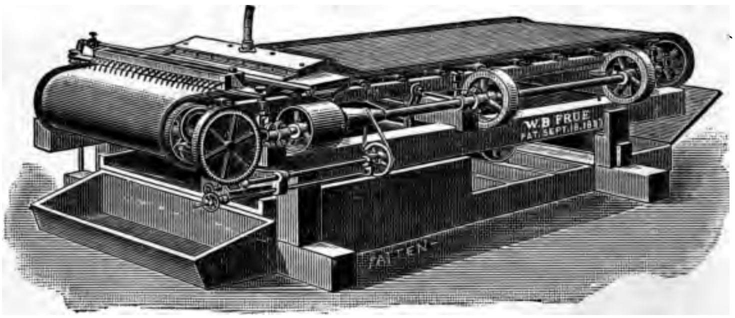

FIG. 37. THE FRUE CONCENTRATOR.

the Frue, Triumph, Woodbury, Tulloch, Embrey, and Johnston are representatives. To produce the best results on these machines, all the stuff should be *sized*.

*The Frue Vanner* (Fig. 37), which has the largest representation in California gold mills, has been frequently described.† It has a side shake of 1″, with from 180 to 200 strokes per minute, the belt traveling upward on an incline from 3′ to 12′ per minute. The belt is made in two sizes, 4′ and 6′ wide, and in the latest patterns as made at the Union Iron Works, San Francisco, has practical arrangement for easily chang-

* From the records of the United States Patent Office.

No. 61,426, January 22, 1867. T. D. & W. A. Hedger, Meadow Lake, California. "Revolving sluice for saving metals."

* * * "The endless apron is made of fabric sufficiently coarse to retain the heavier particles which it receives from the feed spout, beneath which issues a stream of water." * * *

Claim 3. * * * "Separating the ore by passing the valuable portion up the incline and the debris down to the foot, as waste matter, as described." * * *

No. 66,499, July 9, 1867. George Johnston and Edwin G. Smith, Auburn, California. 'Amalgamator and concentrator."

* * * "The pulverized ore or tailings passes to an endless traveling and shaking canvas belt, which ascends against a stream, carrying the heavier particles to be discharged into a box, while the lighter ones are carried off." * * *

Claim 1. The revolving belt or apron (F), with its raised edges (O), having a shaking or rocking motion from side to side, substantially as used for the purpose herein described.

No. 239,091, March 22, 1881. Judson J. Embrey, Fredericksburg, Va. "Ore concentrator."

† See VIth Report of State Mineralogist, p. 92, article on Concentration, by J. N. Adams, E.M.; and VIIIth Report, p. 718, "Milling of Gold Ores," by J. H. Hammond, E.M.

*ing* the slope at the upper end. The frames of these modern styles are made of iron instead of wood. The pulp is discharged very evenly over the belt from a distributor near the upper end, just below the point where clear water is discharged in fine jets across the belt. In placing the machine, attention must be given to the solidity of the frame, and that a perfect level be obtained across the belt; further, the pulp and clear water must be distributed in an even depth of about ¼"; the grade and upper travel depend on the fineness of the pulp, and must be regulated accordingly.

The following guide for a proper condition of the work on the belt is given by Henry Louis, E.M., F.G.S., etc., in his very useful work, "A Handbook of Gold Milling," 1894, p. 324: "The working conditions should be so adjusted that a small triangular patch of sand should show at each of the lower corners of the belt. These sand-corners should not be too large, but must be well marked, and the two should be of equal size. Should they be unequal the fault will be found to be either in that the belt is not accurately level across, that the distributor

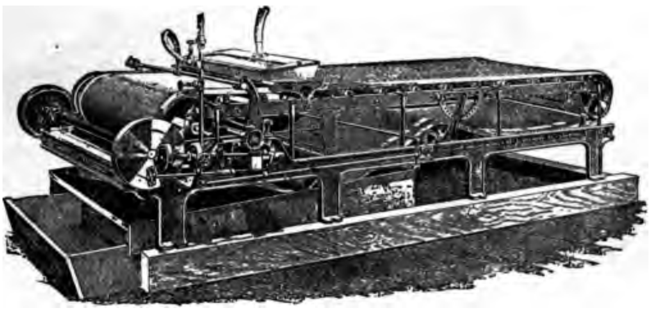

FIG. 38. THE IMPROVED TRIUMPH CONCENTRATOR.

is not doing its work properly, or that some of the working parts have not been properly tightened up, so that there are other motions than the normal ones communicated to the belts. Too large a corner of sand shows that the pulp is too thick, while absence of any corner indicates that it carries too much water."

Two of the 4′ belt vanners, or one of the 6′, handle the pulp from a five-stamp battery. The amount of clear water required to be added is about ⅕ cu. ft. per minute; the vanner requires about ½ H. P

*The Triumph* differs from the Frue, principally, in that it has an end shake of 1″ and slightly quicker stroke (230 per minute), the belt making a forward movement of 3′ to 4′ per minute. It receives the pulp in a bowl containing quicksilver before reaching the distributor, which is all kept in agitation by revolving stirrers.

*The Woodbury* is similar to the Triumph in extent and number of motions, but divides the belt into seven longitudinal partitions; an increased output being claimed for this construction.

*The Tulloch* gives a rocking motion from a fulcrum on the floor, making 140 shakes of 1⅝″ per minute, using either canvas or rubber belt. This machine, it is claimed, saves a somewhat larger amount of the finer and richer grade of sulphurets as compared with the former types.

*The Embrey* is similar to the Frue, but with end shake.

*The Johnston*, with improvements, and the latest of the belt concentrators placed on the market, claims many points of advantage. It is suspended from four non-parallel hangers capable of adjustment, by which the angle of oscillation can be changed as required, preventing the accumulation of sand at the edges, such as occurs with the horizontal side-shake machines, or the piling of the sands in the center of the belt, that occurs with the rocking motion. The motion imparted to this belt resembles more nearly that of the batea than that of any of the other concentrators. The belt is made of No. 6 duck, oiled and painted, but a rubber belt can be used at one third the cost of those with molded edges, which are short-lived. Small, hollow, brass, side-rollers on the shaking-frame, form the raised edges by curving the flat belt slightly upwards. The pulp is delivered from five slots running parallel with

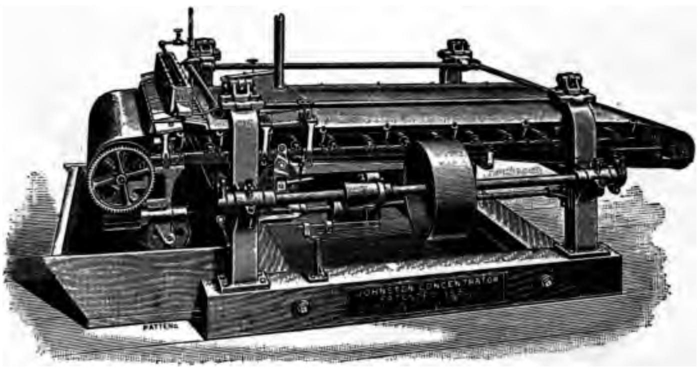

FIG. 39. THE JOHNSTON CONCENTRATOR.

the belt frames, $\frac{1}{4}''$ wide and 16″ long, leaving 10″ spaces, into which the pulp is thrown when it strikes the belt. Here the separation at once takes place; the sulphurets settling on the belt are carried by it up to the clear water, while the sands are carried down the belt. In neither case are the sands or sulphurets obstructed by the falling of water and sands, as in other machines where the pulp is discharged across the belt. The clear water at the head of the table, instead of being discharged from a stationary box to the moving table, is discharged from a distributor, which is attached to and moves with the table, thus stripping the belt of the smallest possible portion of sulphurets. Two widths of belt, 54″ and 72″, are used, which are given a grade of $\frac{1}{8}''$ to $\frac{1}{4}''$ to the foot, making about 118 side-shakes per minute. One machine handles the pulp from a five-stamp battery.

Another vanner, soon to be placed before the mining public, consists of the essential features of the vanner, but carries a rubber belt with depressions all over it, 2″ in diameter and $\frac{1}{8}''$ deep, shaped after the batea, while the entire belt receives a motion corresponding to that given to a batea.

As the motion and grade given to any of these machines can only be correct for a certain size of grain in the pulp, it would be advisable to introduce some method of sizing the pulp previous to bringing it on the

concentrator, and feeding the sized material to different machines. The finer the screen that has been used in the battery, however, the less does the lack of sizing affect the product from the concentrators. The concentrators should always, where possible, be attached to power independent from the stamps, and be placed on a floor below the aprons and in a position to permit the attendant to pass all around and to conveniently transport the concentrated stuff to the covered drying floor, which should be made with a slight incline, preferably of concrete, and exposed to the sunlight.

*Canvas Platforms or Tables.*—Investigation proving that the slimes passing off with the waste from the mill and concentrators still carried an appreciable amount of precious metal, millmen during the last few years have extended their operations, and re-treat the hitherto escaping slimes. This is done by conveying all the waste material from the mill, through sluices, to canvas platforms having the following general features. (See illustrations in chapter on "Typical Mills," pp. 66, 69, and 75.)

A platform is built of clear, seasoned, and planed, 1¼" planking, on a solid, level foundation, and given a grade of about ¾" to the foot, over which No. 6 canvas is stretched smooth, longitudinally, though sometimes crosswise, with a 2" overlap. Particular attention must be paid that the canvas is stretched smoothly and evenly and that no crack opens between the planks constituting the platform. The length and width of the platform required, depends on the amount of pulp to be handled; overcrowding must be avoided. The platform is divided longitudinally into sections corresponding to the width of the canvas, which is 22"; the partition is made of wooden strips, 2" wide and ½" high, covering 1" on the edge of two adjoining pieces of canvas. Running along the head of the platform are two sluices, one placed above the other; one containing clear water, the other pulp from the mill, both furnished with ¾" to 1" plug-holes over each section. Below the lower edge of the platform are two sluices placed side by side, the inside one to convey the waste, the outer one for the concentrates (sweepings) from the platform. When ready for operation, the plugs are withdrawn, and both pulp and clear water commingled flow down in an even current and are discharged through the bottom waste sluice. After one hour or less, the plug is inserted in the pulp-box over the first section, and the clear water permitted to run for a few minutes longer, during which time quartz sand may be observed passing off the canvas, leaving a dark, partly metallic-appearing sediment on the canvas. A tray or board is then placed over the waste sluice, connecting the lower edge of the section with the outside sluice, and the sediment is removed from the canvas, either by sweeping or with the aid of a hose with a flattened nozzle, to be worked later by chlorination or cyanide process.

The following is a description of an improved canvas plant erected and operated in Amador County, by the patentee, Mr. Gates. In this case, the pulp and waste water are conducted from the mill in a flume to the plant, and there divided into two equal streams by the insertion of an adjustable division plate in the flume. The divided pulp passes into boxes (see Fig. 40) 4' long and 1' wide, and having steel screen bottoms with ⅛" and $\frac{1}{16}$" perforations, set on a reversed grade of 6" to the box. The object of these screens is to prevent any chips, leaves, lint, or

foreign substance from passing into the sizing-box (Fig. 41) beneath, which consists of a wooden V-shaped trough, 6′ long, 15″ broad at the top and 2″ in the bottom, constructed of 1½″ boards. A piece of canvas is tacked on the bottom for packing; underneath is nailed a piece of scantling, 4″x6″, at one end of which, reaching within 2″ of the end of the box proper, a slot, 14″ long and 2″ broad, is cut; here a flattened, galvanized-iron funnel, ending in a 2″ pipe, is attached. The pulp falls through the screen with some force and is considerably agitated in the

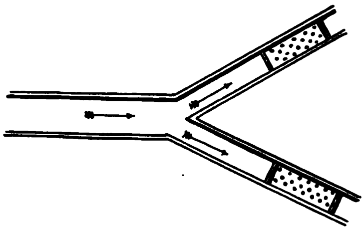

Fig. 40.

separator-box. Naturally the coarser and heavier particles have a tendency to settle toward the bottom. Were the outlet there large enough, all the pulp would pass down and out. Its size of 2″ causes the box to fill to the height of a sluice-box in the end, through which the finer pulp flows to the canvas-tables. To facilitate the separation, a device is placed in the lower end, consisting of an iron pipe, ½″ inside diameter, connected with the main pipe above the screen, and divided into two sections, which are connected by rubber hose for ready detachment. The lower 6″ of the iron pipe has small perforations, through which clear water is ejected, causing an agitation of the pulp. The end of the pipe is stopped with a wooden plug, easily removed. The agitation at the end of the pipe causes the fine material to be carried upward and into the sluice

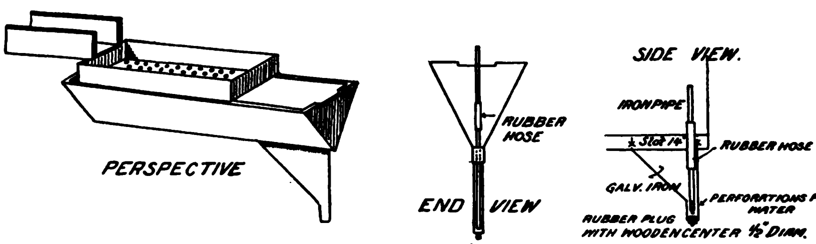


Fig. 41. Sizing-Box.

at the end of the separator-box. Only coarse sand passes through the bottom pipe, and on examining this with a magnifying glass, very few particles of sulphurets are discernible. This separator works well, and disposes of a lot of coarse, valueless material that would otherwise interfere with the subsequent working of the slimes on the canvas platforms. The fine pulp flowing from the top of the separator is conducted in a sluice to a broad, flat box, in which the stream is divided by partitions into ten separate currents, each terminating over a canvas-table, ten in a

row. The pulp goes over a spreader made of strips of galvanized iron, $\frac{3}{4}''$ in height, radiating from a common center to the farthest side of the table, which is 12′ wide. These strips are nailed to an inclined board extending across the canvas-table, having an iron strip, 1″ high, fastened to the lower end, perforated or notched, with indentations $\frac{1}{16}''$ deep and 1″ long, affording a perfect distribution. Twenty tables are arranged in two rows of ten each, covered with canvas laid crosswise and overlapping about 2″. These tables have a grade of $1\frac{1}{2}''$ to the foot, are 13′ long and 12′ wide. After receiving the flow for an hour, it is shut off from the table and a flow of clear water turned on, which in a few minutes washes away the sand, when it is also stopped; then with a hose ending in a flat nozzle, the accumulated sulphurets are washed from the canvas into a trough below, extending along the base of the entire series. In order to secure sufficient fall for this sluice, each succeeding table is set 4″ lower than its predecessor, giving 40″ fall on 125′ of sluice length. Two extra tables are arranged, one at the end of each row, to take up the surplus flow during the time one of the tables is shut off, to avoid overloading, as each table already carries the proper amount of pulp. The effectiveness of the canvas-tables depends on maintaining an even flow of pulp during a given time; it will not do to overload them. All the pulp that leaves the table is considered waste, and is collected in a flume, to be used a short distance off as power on an overshot wheel, by means of which the patentee runs a vanner of his own invention. This waste water is caught up again and used on a second wheel, which also runs a vanner. The sulphurets washed from the tables flow through a sluice to a box outside the building, 12′ long, 2′ wide, and 12″ deep, with a cross-piece 2′ from its upper end, reaching within 2″ of the top of the box; in this upper section the coarser grade of the material is retained, while the finer flows over the weir. The two grades are shoveled out separately and placed in separate V-shaped boxes, over which are perforated iron pipes, from which small streams of water trickle, gradually carrying the pulp down and passing it through sluices onto the spreaders of separate vanners. These two machines work with different motions, doing excellent work on this impalpably fine stuff. The slimes flowing from the washing-boxes beneath these vanners are conducted, with the overflow of the two compartment boxes above referred to, to two other canvas-tables, below which they are allowed to escape as waste; not that they have given up all the precious metal they carried, but because the point is reached where it is more economical to lose the remnant than to attempt to save it.

As the slimes from most of the canvas plants, as usually operated (especially where the ore crushed carries a heavy percentage of sulphurets, or has been stamped with a high discharge), are still valuable in gold, they can be conveyed to so-called slime-settlers, or tanks. These tanks, for there are generally several, are placed below the canvas platforms, and are about 2′ deep, 2′ wide, and 12′ to 20′ long; they are divided into sections of 2′ square, by 2″ plank set on edge, extending alternately from each side, leaving an opening 4″ wide and 2′ deep, causing the slime water to take a serpentine course in passing through. The tanks stand level, and the slimes, in settling, form their own grade as they enter at one end of the tank, and, passing through the succes-

sive sections, issue at a diagonally opposite point only slightly clouded. These tanks require cleaning only at long intervals.

Up to the present time, the concentrates in the California mills have been generally handled by the chlorination process, to free them from their gold, but within the last year several plants are successfully working them by the cyanide process.

The tendency in the construction of mills at the present day is to a substitution of steel for iron, where possible, and to an increase in the weight of the stamps.

A greater application of grinding and amalgamating machines, in place of or subsidiary to the stamp-mill, is also noticeable, the most popular of which will be shortly described.

For a more thorough appreciation and knowledge of the work done by mills, records should be kept, by the amalgamator, of all transactions connected with mill work, showing every item, loss of time, consumption of mercury, iron, fuel, water, amount of rock treated, etc., in addition to the records kept in the assay office. This is already being done to some extent, but such records should be kept in the small mills as thoroughly as in the large ones.

### GRINDING AND AMALGAMATING MACHINES.

*Arrastras.*—Although the arrastra has been largely superseded by the stamp-mill, the fact remains that it is the best and cheapest all-round gold-saving appliance we have. Hence, its use is always indicated where small, rich veins are worked in the higher mountain regions, but it is also found valuable placed below the present quartz mill, where the waste waters from the mill can be picked up and used over again for power on horizontal or overshot wheels. In these cases, it handles the tailings from the mill after they have passed over the concentrators and canvas-plants. This part of the milling is usually leased to parties who pay the mine a fixed amount per ton for the tailings, the lessees putting up all their own machinery. These arrastras are built of a size to handle at least 4 tons of tailings in twenty-four hours. Their foundations are either formed of hard rammed clay, concrete, or a plank platform with broken joints, on which a bed of clay is placed. The foundation is always made larger than the circumference of the proposed arrastra. The bed is formed of rocks harder than the substance to be crushed, usually fine-grained basalt, granite, or quartzite. These are picked with a partially level surface, and as near of the same thickness as possible, usually from 1′ to 2′ thick. They are built around a center cone, forming an annular ring from 2½′ to 6′ wide, and are laid with narrow spaces between each rock, into which dry clay should be tightly rammed to within an inch of the surface. The outer circle is formed of rocks or staves, with rammed earth behind, and built from 2′ to 4′ in height. On the central cone, which consists of stone or a block of wood, and which stands somewhat above the paved bottom, a center post is let in, from which project four arms at right angles to each other, and extending nearly to the outer circle. Heavy, hard rock-drags, weighing from 200 to 1,000 lbs. each (from 400 to 600 lbs. is the usual weight), are attached to the arms by ropes or chains passing through eye-bolts secured in the rock-drags. They are placed so that part of them drag near the cone, with the inside corner slightly in

HORSE-POWER ARRASTRA, KERN COUNTY.

WATER-POWER ARRASTRA, KERN COUNTY.

STEAM-POWER ARRASTRA, KERN COUNTY.

advance, while the remainder sweep near the outer circle with the outer corner in advance. The front edge should always be slightly elevated, so as to permit of the particles passing under the drag instead of being pushed ahead.

Where a horizontal wheel is used, the arms are attached to the center post and the wheel encircles the arrastra, the water striking on buckets set to an angle of 45°. With overshot wheels the arrastra may be run by a belt and pulley attached to the center-post, or by a spur gearing. It requires about 6 H. P. to run an average-sized arrastra. Running tailings, a speed of 15 to 30 revolutions per minute is given; crushing ore, the arrastra should be run slower and the pulp thicker.

For discharging the arrastra, plug-holes at different levels are put into the outer circle, leading the pulp into sluices lined with plates, riffles, and blankets. In some cases the arrastra has been made to work continuously by fitting a screen to a part of the outer circle and letting it discharge into a line of sluices. As the arrastra bottom and drags are extremely uneven and rough when first set up, some coarse sand and water are introduced on first starting, and the drags are allowed to run slowly until somewhat smoothed down, before the regular charge is introduced. The machine is usually only cleaned up thoroughly when the bottom is worn away; between times the crevices are picked out for the depth of an inch or two with picks, scrapers, and spoons, and panned out, with what pulp remains on the bottom, after the charges have been successively thinned down and run off through the plug-holes. If crevicing has been done, a little fresh clay can be rammed in to within 1″ of the top of the bed. During the grinding of the charge, the quicksilver is introduced through a cloth; the amalgam should be kept drier than in the stamp battery, though not sufficiently so as to become "crumbly." Great attention must be paid to tamping the bed in solid, otherwise an excessive loss of quicksilver may occur. Continual horn tests of the pulp furnish a guide for the proper working.

Machines have, from time to time, been introduced in California to replace stamps, claiming to do more effective work, both as regards the crushing as well as the amalgamating. Those mostly seen in operation, and finding the most favor, are the Huntington and the Bryan mills, which may be taken as types, and which reduce the ore by a continuous rolling motion; in the one case the roller acting on a ring on the circumference, and in the other on dies in the bottom.

*The Huntington Mill* consists of a shallow iron pan with a central cone, through which an iron shaft revolves. Bolted on the sides of the pan and inclosing it, are semi-circular iron sections made in two halves and also bolted together; one of these sections contains an opening about 9″ deep, divided into three parts, into which curved iron screen-frames are keyed, while the other section contains a feed-trough, attached near the top. Between the bottom of the pan and the lower edge of the screen-frames an iron or steel ring-die fits against the sides of the shallow pan, being secured by wooden wedges; against this die, four rollers, suspended from yokes resting on an iron cover, revolve, receiving their motion from the central shaft. These suspended rollers are pressed by centrifugal force against the ring-die. Each roller is encircled by an iron or steel shoe fastened by wooden wedges; this can be renewed when worn too thin, or when it becomes unround—flattened. Means

are provided for lubricating the shafts on which the rollers work, without permitting the lubricant to come in contact with the pulp. As the rollers hang about ½″ above the bottom of the pan, scrapers are attached to the revolving cover between the rollers, and reaching to the bottom of the pan, to prevent the baking of the pulp.

The size of the pan most frequently used is 5′ in diameter, though for prospecting purposes one of 3½′ is also made; the former is run at a speed of 70 revolutions per minute, the latter at 90 revolutions. They are provided with self-feeders, which introduce the ore at regular intervals—the only way in which they can be operated, though not correct in

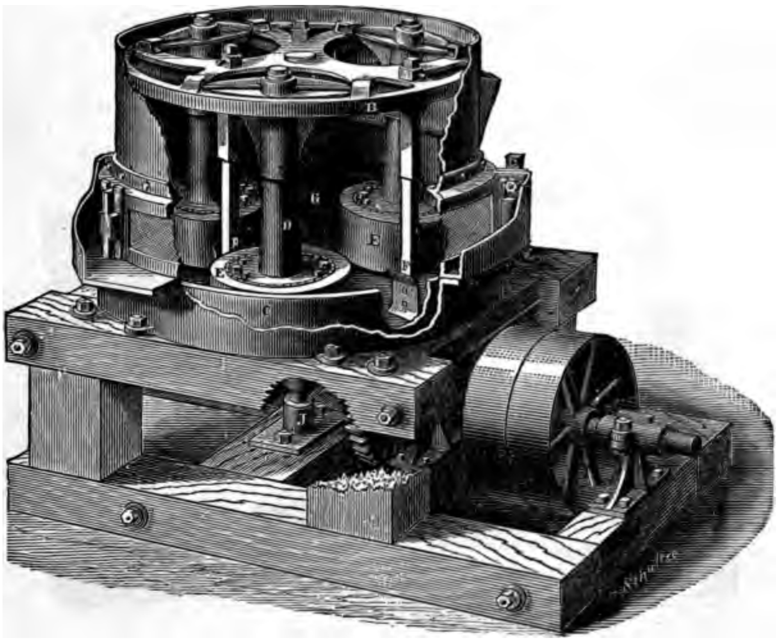

FIG. 42. THE HUNTINGTON MILL.

principle. A 5′ mill requires about 8 H. P., and crushes about 20 tons per day. Before starting up a certain amount of quicksilver, up to 50 lbs., is introduced into the pan with some water and rock. The supply should be regulated to make a stiffer pulp than in a stamp-battery; quicksilver is added from time to time. A groove in the bottom of the pan, connecting with a plug-hole on the outside, permits of the quicksilver and amalgam being drawn off at intervals to recover the latter, after which the former is returned. If the pan is working correctly the bottom around the center remains bare; this can be observed through the cover while running; when not bare, it is a sign that the pan is being overfed. As the machine throws the pulp with considerable violence through the curved screens, a shield is placed outside of them, directing the pulp into a narrow sluiceway with a spout opening on the apron-plate. It is claimed that the percentage of gold amalgamated and saved on the inside is far greater than in the stamp-mortar, going above 80%; all rusty gold being subjected to a heavy scouring action.

The Russian-iron screens used are short-lived; they can be made to last somewhat longer by placing a false screen, made from an old worn screen with the openings enlarged, between the pulp and the screen proper.

Great care must be exercised in putting up one of these machines, to get it perfectly level and on a rigid foundation, and to keep all the bolts holding the pan on the foundation well tightened up; the feed also requires close observation.

When cleaning up or renewing the ring-dies or shoes, the top cover, with the suspended rollers, are lifted out with chain block and tackle, leaving the interior of the pan free for operation.

The mill works well on soft quartz and clayey ores, introduced in pieces not larger than walnuts. A great drawback to the machine is that the rings on the rollers and also the dies become "unround," so that instead of rolling smoothly, they strike in places, necessitating changing the rings before they are worn out; this changing takes up some time.

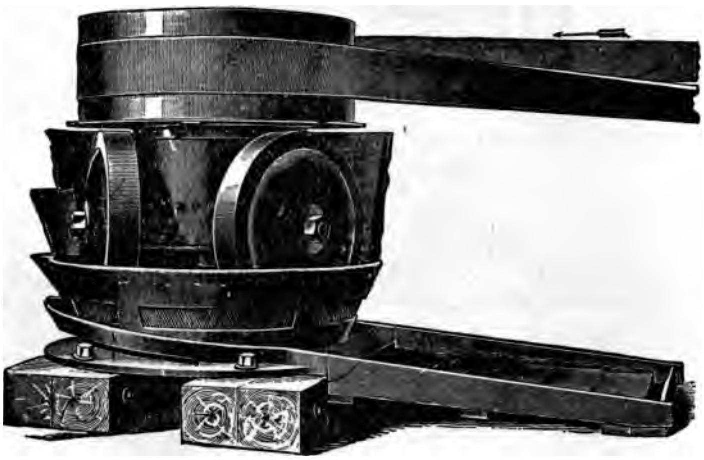

FIG. 43. THE BRYAN MILL.

The opinions of millmen who have handled the Huntington mill, as to its merits, are very diverse. Where the ore produces a large amount of fine stuff, by using a grizzly with closely set bars, the Huntington can be run to advantage on these "smalls" in conjunction with the stamps.

*The Bryan Roller Mill* is a modified form of the Chili mill, built in sizes of 4′ and 5′ diameter. It consists of an annular mortar with an outside gutter and spout, cast solid, containing steel dies arranged in the track of three crushing rollers, which in the 5′ mill have a crushing-face of 7″, a diameter of 44″, and weigh 3,650 lbs. They have fixed axles, "journaled" in a central revolving table, attached to and driven by a belt pulley. This pulley is a cylindrical tank, which, in the smaller pattern, rests immediately on the rollers, and can be made to increase their crushing power by being loaded. The mortar is supplied with curved screen-frames around the entire machine, the pulp being discharged all around into a gutter delivering through a spout, on one side, to an apron-plate.

The chief wearing parts are the steel dies and tires on the rollers; these latter are fastened to the rollers by wooden wedges. According to the statement of the manufacturers (Risdon Iron Works, San Francisco), one set of these wearing parts will crush from 4,000 to 8,000 tons of ore in the large size, and 1,500 to 2,000 tons in the smaller size, and at the rate of 25 to 35 tons and 12 to 20 tons per day, with a speed of 30 and 60 revolutions, respectively, per minute, the smaller size requiring from 5 to 6 H. P. The oil channels for lubricating the bearings are arranged to prevent the oil from entering the mortar. To keep the pulp

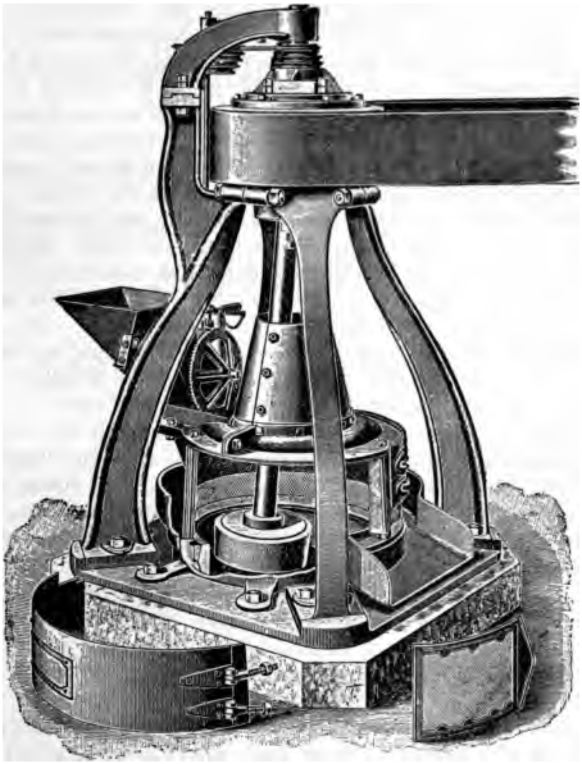

FIG. 44. THE GRIFFIN MILL.

from baking to the rollers or dies, and to assist in equalizing the ore received from the feeder, scrapers with adjustable springs follow each roller. They are also provided with self-feeders. In operating the mill, ore, water, and mercury are introduced into the mortar, the pulp passing around next the screens in a current not less than 300′ per minute, while the motion inside of the rollers is much slower. The amalgam, working its way toward the center cone, is kept from being re-ground, and can be observed while the mill is in operation; it is claimed to retain 80% of the amalgam in the mortar. To clean it up, the dies between the rollers are removed, the pulp and amalgam taken out, and wooden blocks of the thickness of the die put in their stead, on which the rollers are revolved, when the remaining ones can be taken up. It is claimed for these mills,

that they wear smooth, and even while crushing hard quartz, discharge freely (on account of large screen area), avoid sliming and flouring of quicksilver, are good amalgamators, can be cleaned rapidly, are easily put in place, and require small power for amount of work done.

*The Griffin Mill* belongs to that class of mills using a roll running against a ring or die; but instead of several rollers, as in the Huntington, this has one roller only, swinging from a longer shaft, hung from a point in the central axis of the mill, and rotated about its own axis by the power applied at the top. It is run at a speed of 190 to 200 revolutions per minute, crushing from 1½ to 2½ tons per hour, the power being applied to a horizontal pulley above, from which the shaft is suspended with a universal joint, and the roller is rigidly attached to the lower extremity of the shaft. The roller swings in a circular pan supplied with a ring or die, against which the roller works; and carries on the under side scrapers or plows to prevent the pulp from baking. A circular screen-frame is fastened on the pan, to the top of which a conical shield is attached at the apex, through which the shaft works. The pulley revolves upon a tapered and adjustable bearing, supported by the frame composed of iron standards, two of which are extended above the pulley to carry the arms in which is secured the hollow journal-pin. The shaft is suspended to a universal joint within the pulley. This joint is composed of the ball or sphere with trunnions attached thereto, which work in half boxes that slide up and down recesses in the pulley-head casting. The lubricant is supplied, for all parts needing it, through the hollow pin. The roll revolves within the ring-die in the same direction that the shaft is driven, but on coming in contact with the die, it travels around the die in the opposite direction from that in which the roll is revolving with the shaft. A pressure, by centrifugal force, of 6,000 lbs. is brought to bear on the material being pulverized between the roll and die. The water is introduced with feed when running, and receives a whirling motion from the roll, which brings the pulp against the screens, 9′ in area. A circular trough on the outside of the pan conducts the pulp to one side, where it discharges over an apron.

## TYPICAL CALIFORNIA GOLD MILLS.

As the details in milling practices of the several counties of the State vary greatly, the following typical mills have been selected to indicate the practice under varying conditions:

No. 1. *Amador County.*—The ore is a soft, easily crushed quartz, with about 1½% sulphurets, and is largely mixed with slaty material, which, to the extent of 25%, is found mixed with the concentrates. The stamps weigh 750 lbs. each, and drop 6″ about ninety-five times per minute, discharging through a No. 8 slot screen, at the rate of 2½ tons per stamp in twenty-four hours. The stamps drop in the following order: 1, 2, 3, 5, 4; Nos. 1 and 2 having ½″ more drop than the other stamps; in the adjoining battery the order is reversed. Iron shoes and dies are used. There is an inside plate used in the battery, which retains about 75% of the amalgam. The apron is 48″x13″, set on a grade of ¾″ to the foot, and the double sluices below are 9′ long by 14″ wide, with a grade of 1½″ to the foot. From these sluices the pulp

passes to vanners. To clean the sulphurets from the slaty admixture, a cradle, 12′ long, 20″ wide, and 4″ deep, has been placed in the mill, run by an eccentric. The dirty slimy sulphurets are taken from the washing-boxes beneath the vanne and placed in a half barrel standing on the floor of the mill, into which a hose is lowered, and the sulphurets

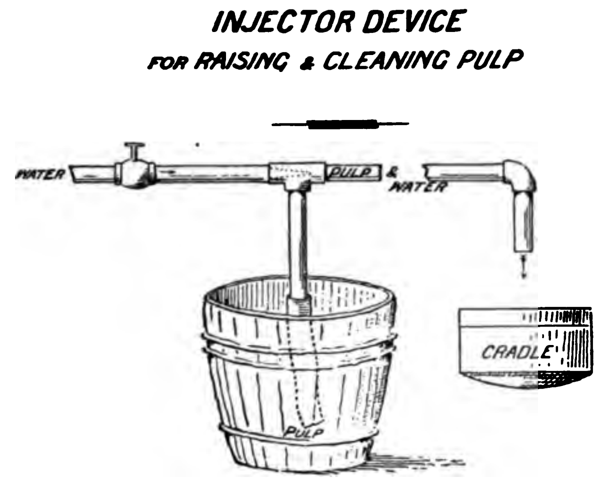


FIG. 45.

are raised from the barrel to the cradle by creating a vacuum, through a small jet of water under pressure forming an ejector. The pulp in the cradle is stirred vigorously toward the head; the grade is from 7″ to 8″ in 12′. This washing in the cradle relieves the pulp of about 25% of waste material. Twelve tons can be washed in a day. The canvas-plant below the vanner has some interesting features. The canvas strips are only 12″ wide. The pulp as it leaves the vanner is carried to a mercury-trap, consisting of a box of diminishing width, with three

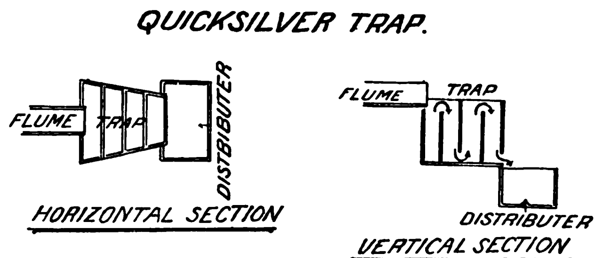


FIG. 46.

upright divisions, under and over which the pulp flows. From the mercury-trap the pulp falls into a long box, about 1′ square at the ends, in the bottom of which are ten holes, whose size is regulated by experience; they must equalize the discharge with the inflow from the mercury-trap. The pulp introduced into the long distributor-box sizes

itself to some extent by gravity; the finer material, being held in suspension longer, finds its way out at the end of the box, while the coarse quickly passes through the holes in the bottom, nearer the center of the box.

There are twelve strips of canvas, 100′ in length, each strip having a width of 12″ and a grade of 4½″ in 12′. The coarse material is all found

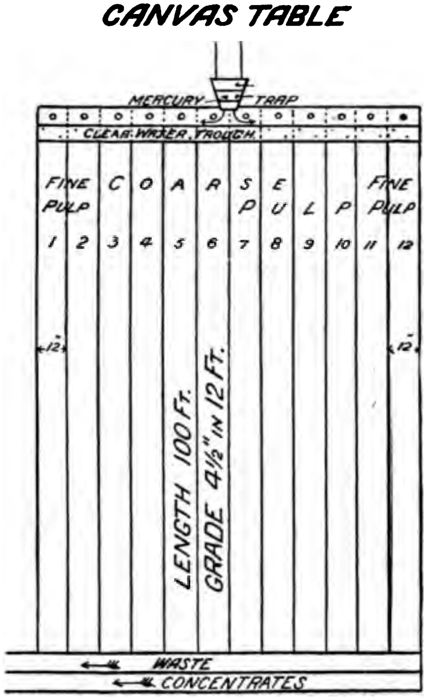


FIG. 47.

on the six center sections, the two outside sections on each side carrying the finer material. An additional series of tables, with 20″ wide sections and a grade of 9″ in 12′, receives the pulp after passing over the first.

No. 2. *Amador County.*—The practice of this mill in handling their tailings may be taken as an example of the better methods now practiced in the State. This mill has 900-lb. stamps, dropping 85 times per minute, with a 6″ drop and a 7″ discharge, kept constant by the use of lower chock-blocks. No. 30 brass-wire screens, 4′ long and 4″ wide, set vertical, are used, giving a duty per stamp of 2½ tons in twenty-four hours. The batteries are supplied with inside front plates. The apron-plates are 46″x30″, set on a grade of 1¾″ to the foot. These are followed by 18′ of sluice-plates, 15″ wide, the first 10′ of which are double. About 66% of the amalgam is recovered in the battery. The loss in quicksilver, which is introduced into the battery every half hour, amounts to about 1½ cents per ton. The total cost of milling at these

works is given as 70 cents per ton. The mill is supplied with three vanners to each battery, with 4½′ belts. The pulp from the plate-sluices passes directly to the spreaders of the vanners, a division into thirds being first effected. After leaving the belts, the pulp flows through sluices to a flume, where it is divided into two equal streams by the insertion of an adjustable division plate in the flume. The divided pulp passes to two steel screens with perforations of ⅛″ and $\frac{1}{16}$″ respectively, which form the bottoms of two 4′ boxes, 1′ wide, set on a reverse grade of 6″ in 4′. These boxes prevent any foreign substance from passing through into the sizing-box below, and clogging the outflow pipes. After the passage of the screens the pulp falls into a separator, consisting of a wooden V-shaped trough, 6′ long, 15″ wide on top and 2″ at the bottom, with a flat, funnel-shaped discharge pipe of galvanized iron attached at one end, ending in a round 2″ pipe. As more pulp enters the separator than can be discharged through the 2″ pipe, it fills and flows over the end into a launder; the heavier and larger particles sinking down and passing through the pipe. The overflow passes on a spreader that delivers it to a canvas-table, with ten sections; a second similar table, placed below, receives the waste from the first one. The tables are 12′ wide, 13′ long, and set on a grade of 1½″ to the foot, and to secure a proper grade for the waste-sluice, each section is set 4″ below its predecessor. All the waste water passing from the tables is used a short distance off as power on an overshot wheel that runs a vanner, on which are worked the concentrates taken from the tables.

No. 3. *Butte County.*—The quartz carries considerable sulphurets. When hoisted from the mine it is dropped immediately over a grizzly, with the bars placed 1½″ apart; the coarse rock crushed is loaded into cars, and trammed to the mill, distant about 150 yards, and dumped into bins which are calculated to carry 1,500 tons. From here chutes convey the ore to the Challenge self-feeders. These are operated from the center stamp in each battery. The stamps, which are supplied with steel shoes and dies, weigh 850 lbs., drop 7″, and about 100 times per minute; the discharge is 7″; the screen is No. 8 diagonal-slot, 8″ wide; each stamp crushes 2½ tons per twenty-four hours. The screens, which last about four weeks, are used later in the chlorination works for the recovery of cement copper. From the mortar the pulp passes over a 14″ mortar plate; thence to a 4′ apron and 12′ of sluice-plates; aprons and plates are set to a grade of 3″ to the foot. The pulp then passes over the vanners, two for each battery, after leaving which, it is conveyed to the canvas-platform house. The canvas-platform is 24′ wide and 60′ long, covered with x 2 0 0 canvas, and below it are 150′ of settling-boxes. The plates are scraped every day, and dressed besides, when required.

No. 4. *Calaveras County.*—The rock consists of massive quartz, schistose and slaty diabase, and chloritic and talcose schist, with iron sulphurets; it is crushed in jaw-breakers at the head of the shaft, after passing over grizzlies, and is dropped into bins, from which the ore is conveyed, in cars, to three other bins in the mill, one for each section of twenty stamps, having a capacity of 600 tons each. The bins discharge into Challenge self-feeders. The sixty stamps weigh 775 lbs. each, and drop 105 times per minute, the drop being 6″, and the discharge 10″

from the new die. Only one chock-block is used, causing the height of discharge to constantly increase. The duty of the stamps is 4 tons in twenty-four hours. Round-punched tin screens, 10" x 14", are used. They are lightly burned before using. Three and a half of the screen sheets are tacked on the screen-frame on three sides; the top side is secured by a long, narrow strip of wood screwed to the frame. The superficial area of the discharge is about 287 sq. in. The screen-frame is braced by six cross-ribs, to which the screens are tacked. A splash-board is suspended in front of the screen by eyebolts and hooks, with a strip of canvas tacked along the bottom, the full width of the screen. An iron apron or table is secured to the front of the mortar below the screen, the bottom of which falls 1" below the lip of the mortar, permitting the insertion of a rough inch board, 9" in width, in front of the mortar, flush with the upper edge of the lip of the mortar; on this the

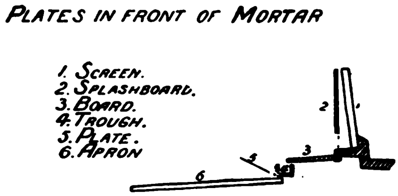


FIG. 48.

pulp falls from the screen, and it is claimed to be superior to a plate in retaining the amalgam. Three inches below the board, runs a trough, in which are two apertures one third the distance from each end, which allows the pulp to fall on a short, 6" wide copper plate with a pitch toward the mortar, and from thence to the apron-plate, 2' wide and 24' long, set to a grade of 2" to the foot. An inside front plate is used in the mortar. From the apron-plate the pulp passes to a sluice-box and is conducted to the spreaders of the vanners, of which there are twenty-four. After leaving these, the pulp is led through a sluice-box and flume one mile long to the canvas-plant. The plates are dressed every morning; a battery is hung up, the water shut off, the splash-board removed and washed off, as also the screen and the entire front of the battery, to remove all sand; the plate is then vigorously scoured with a whisk-broom to loosen the amalgam. A very dilute solution of cyanide of potassium is sometimes used during this operation, and the loosened amalgam brushed to the foot of the plate. The plate is then scraped upward with a piece of rubber 4" x 4" and ½" thick; a piece of rubber belting would answer the same purpose. The collected amalgam at the head of the plate is removed in a scoop and placed in a safe. The plate is then sprinkled lightly with quicksilver, which is spread evenly over the entire plate, the water turned on, and the stamps dropped. The operation for all the plates requires nearly three hours.

A clean-up of the mill is made monthly or semi-monthly, according to the condition of the battery amalgam, at which time all necessary repairs are made, and new shoes and dies are fixed if required. Shortly before hanging up, the feed is shut off to permit the ore to be crushed down as low as possible. The water is then shut off from the battery, splash-board and screen removed, and all hosed off; the inside plate is

moved, and the amalgam scraped off The contents of the battery e now removed, and placed in the revolving clean-up barrel, the dies placed, tappets set, screens replaced, and the mill started. The pulp at leaves the mill carries considerable auriferous pyritical slimes, as ight be inferred from the high discharge used in the battery. This is nveyed through a sluice 12" x 8", to a canvas-plant one mile distant. st before entering the building set apart for the recovery of these mes, the sluice is widened to 18", and divided into three sections by o narrow strips fastened to the bottom. These divisions fork off into parate sluices, which are again subdivided. Three of these subdivis-

*DISTRIBUTORS.*

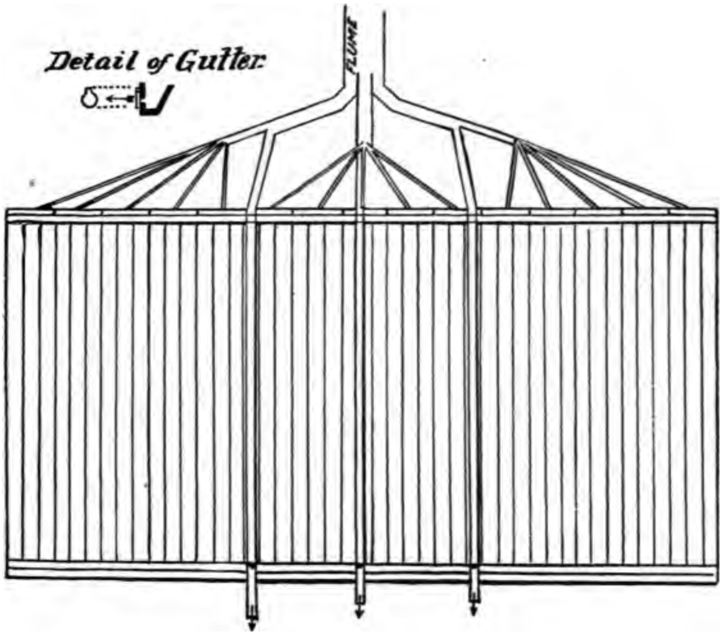


Fig. 49.

ns are carried directly through the building, and there divided, and e other three are divided off in five separate sluices, one for each sec- on of the canvas-table. There are forty-five sections for each table— nety in all. They are 42′ long, 22″ wide, and set on a grade $1\frac{1}{8}$″ to e foot. No. 8 duck canvas is used, and when worn on one side it is rned; it lasts about one year. The last division of the pulp, outside e building, is into five boxes, 4″ square, each of which terminates in receiving-box, reaching across three canvas sections, about $5\frac{1}{2}$′. The e divisions supply one third of the sluices on one side of the build- g; the pulp passing to the canvas through an auger-hole in the side the box. The flow is regulated by a slide suspended over the whole. ove the pulp-distributing box is a clear-water box, and at the lower d of the canvas-tables are two sluices, side by side—one to receive the ncentrates, the other for the reception and discharge of pulp. The rrent must be thinned and distributed so that no accumulations m.

No. 5. *El Dorado County.*—The quartz carries considerable slate mixed with it, and about 3% of iron sulphurets. The stamps weigh 950 lbs. each, and drop 4″ 104 times per minute, discharging through a No. 2 sheet-tin, perforated screen with from 5″ to 7″ discharge, crushing 3 tons per stamp in twenty-four hours. The shoes and dies are both steel. The battery is supplied with an inside plate in front. The apron-plates are 16′ long, set on a grade of 1¾″ to the foot, and are followed by 6′ of sluice-plates, 2′ wide, on the same grade. These plates are dressed every day, but only scraped once a month. (Note:—This is not advantageous, as the constant scouring action of the pulp undoubtedly detaches fine particles of amalgam.) The batteries yield 62% of the amalgam. Eight Woodbury concentrators receive the pulp. The quicksilver is introduced into the battery every half hour; the loss of quicksilver being estimated about one pound to every ten tons of ore.

No. 6. *El Dorado County.*—The quartz carries about 2% of sulphurets and contains slate mixed with it. The stamps weigh 750 lbs. each and are fed by Challenge feeders; no rock-breaker is used. The stamps make 96 drops per minute, varying from 4½″ to 6″, with 7″ discharge. The mortars are wide, and have an 8″ wide inside amalgamated plate; the screen is perforated tin, equal to No. 7. The apron is 54″ by 42″, with grade of 1¾″ to the foot, followed by sluice-plates 12′ long, which are double on one battery and divided into four divisions on the other; this latter arrangement gives better results. Below the sluice-plates is a blanket-sluice, 6′ long and 14″ wide, the blanket being washed twice a shift. From these the pulp passes to the Frue vanners with 6′ belts; the same wheel runs both stamps and vanners. The plates are dressed twice in twenty-four hours, but are not scraped until the clean-up, once a month. One and three quarters tons are crushed to the stamp per twenty-four hours.

No. 7. *Mariposa County.*—This mill is working on ores containing gold in a very finely divided state. There are ten stamps of 900 lbs. each, fed by self-feeders. These stamps drop 96 times per minute, with a 7″ discharge while the die is new; when it is worn down one half, a smaller chock-block is placed under the screen. The pulp is retained in the mortar for a long time. The stamps are only raised to the level of the water as it stands in the mortar; and a front inside plate is used. The screen is a 60-mesh. It is claimed that 70% of all the amalgam saved is taken from the inside battery-plate. There are three apron-plates to each battery; the first is immediately in front of the splash-board, next to the lip of the mortar, 12″ deep and the width of the mortar. This is followed by a 2″ drop onto a second plate 3′ deep, across the width of the mortar, succeeded by a 1″ drop to a 4′ apron-plate. From this plate the pulp passes immediately through distributing pipes to the vanners, of which there are three. Two of these are 4′ wide, taking the pulp from one battery, while a 6′ belt vanner takes the pulp from the other battery. The narrow belt vanner gives the best satisfaction.

No. 8. *Nevada County.*—The ore is delivered by car at the top of the mill into grizzlies, the bars of which are 2¼″ apart, and which deliver the coarse stuff to a crusher of the Blake type, through a bin with chute

immediately over the crusher, keeping the same constantly supplied without the aid of a shovel. From the rock-breaker the ore is delivered into bins with chutes connecting with the Challenge self-feeders. The feed is operated from the center stamp. The stamps drop 7″ and 86 times per minute, with a 4″ discharge. Steel shoes on iron dies are used. The screens are English sheet-tin, perforated, No. 10, five pieces making a complete screen, costing 50 cents, and lasting one month. The steel shoes on iron dies have records of over 300 tons to the stamp, the daily average being from 1¾ to 2 tons per stamp. The plates are divided into an upper apron, 18″ wide, followed by a 4′ apron. Between the two, catching the pulp from No. 1, is a box 3″ wide, with a perforated screen bottom, somewhat coarser than the battery-screen. This acts as a distributor on No. 2 apron, retaining any coarse pieces. For the forty stamps and accompanying concentrators, 15 miner's inches of water are used, all applied on the inside of the battery. The aprons have a ⅝″ grade to the foot. Below the apron are 12′ of sluice-plates, part of them 30″ wide, while the others have the same width as the apron above. These plates are cleaned up every morning with pieces of rubber belting; it takes about fifteen minutes to clean one set. From the sluice-plates the pulp passes over a 12′ shaking-table covered with silver plates; these plates receive their motion from an eccentric placed underneath. Passing the table the pulp enters a box, from which it is conveyed through pipes to the vanners on a lower floor, two for each battery.

No. 9. *Nevada County.*—The stamps weigh 750 lbs. each, drop 6″, 95 times per minute, with from 4″ to 6″ discharge, and crush from 1¾ to 2 tons per day, using steel shoes and dies. There is a 5″ wide silvered plate in the front of the battery. A No. 9, perforated, sheet-tin screen is used; it is not burnt before putting on, and is turned when the lower edge is worn, lasting on an average 30 days. In front of the screen is a splash-board, provided with an 8″ plate next to the screen. The upper apron-plate is 18″ deep, set on a grade of ¾″ to the foot, with 12′ of apron-plates below, divided into four plates of 3′ each, set on a grade of 1″ to the foot. From these the pulp drops into a box running across the end of the plate, from whence it passes to the vanner. The plates are scraped every twenty-four hours, with the exception of the upper 4″ next the mortar, and are dressed twice a day, using dilute cyanide of potassium. Both rubbers and chisels are used in scraping the plates. In cleaning up the batteries, which occurs once per month, the headings are put into a revolving barrel with pieces of iron and quicksilver, and after running several hours, the contents are removed in buckets, the sand "boiled out" with the hose, the dross skimmed off, and the quicksilver strained. About 75% of the amalgam is saved in the battery. The tailings assay from 25 cts. to $1 50 per ton.

No. 10. *Nevada County.*—The stamps weigh 800 lbs. each, and are given a 6″ drop, 100 times per minute, with a discharge varying from 2″ to 4″; there are no plates in the battery. The ore passes over grizzlies, with bars 1½″ apart, to a No. 2 Blake crusher; thence to the ore-bin that supplies the Challenge feeders, which are operated from the center stamp. Steel shoes and iron dies are used—the shoes lasting, on an average, 155 days; the iron dies, 70 days. No. 6 Russian-iron slot-screens are used. The outside mortar-plate is 14″ wide, with ½″ pitch

to the foot, and retains 75% of the plate amalgam. The apron below is 4'x4', with a grade of ¾" to the foot. Beyond this are 12' of double sluice-plates 12" wide, and with ¾" grade to the foot. Three sand-boxes separate the different apron-plates. From the sluice-plates the pulp passes directly to the concentrators. The duty of the stamps is two tons per day. The tailings assay $1 80 to $2 per ton. From 10 to 12 lbs. of quicksilver per month is used for the 40 stamps. The plates are scraped every day, and the batteries cleaned once a month, the headings being worked in a Knox pan. A weak solution of cyanide of potassium is used in dressing the plates.

No. 11. *Placer County.*—The quartz carries but a small percentage of sulphurets, and is delivered from the mine over an incline tramway to two grizzlies with 12 bars, 3" apart, 12' long, 3" deep, and ½" wide, set on an angle of 45°. In front, below, and between the grizzlies is a Blake crusher, from which the ore drops into the bin that supplies the Challenge feeders. These are operated from the center stamp. The stamps weigh 750 lbs. each, and drop 5", 90 times a minute, and the discharge averages 5". The screen is set on a 4" block, with a 5" plate on the inside. The screen is a No. 10, slot-punched, set with a slight incline. Part of the water for the battery is supplied from a small wooden trough, pierced with holes in front of the screen. The outside iron lip of the mortar is covered with a silvered plate. The apron, set on a grade of 1½" to the foot, is 4' long, and is followed by 12' of sluice-plates, 18" wide. After passing through a quicksilver trap, the pulp passes through a 3" pipe to the Frue vanners. A tank of quicksilver is used every three months, in crushing 3,500 tons of ore. The plates are scraped every day with rubbers, and are occasionally dressed with phosphate of lime, or with lye. The battery is cleaned out once a week, and yields 50% of the amalgam.

No. 12. *Plumas County.*—The ore is free-milling, and contains about 1¼% of sulphurets. It is delivered to the Blake crushers in the mill by an incline tramway, and the ore passes through the bins to the Challenge feeders. The stamps weigh 850 lbs. each, dropping 8½", 80 times per minute. The discharge varies from 6" to 8", through No. 8 diagonal-slot punched screens, with a discharging surface to each battery of 45" in length by 6" in height. The mortar is furnished with a lip plate and a cast-iron trough, which receives the pulp, also with a 5" inside plate. The pulp passes from the trough to the apron and sluice-plates, which have a grade of 1¾" to the foot and a length of 20', and is then passed to the concentrators. Below the mill the tailings are picked up by outside parties and re-ground in arrastras. The tailings assay $2 per ton. The loss of quicksilver at this mill is about a flask for every 4,600 tons crushed. The cost of milling does not exceed 50 cents per ton when using water-power. The plates are cleaned every twenty-four hours. About 60% of the amalgam is derived from the batteries, which are cleaned up once a month. The headings are placed in an iron revolving barrel, and the panning-out is done with a batea.

No. 13. *Plumas County.*—The ore is hauled to the mill by wagon, and is broken and fed by hand. The stamps weigh 750 lbs. each, drop 5" to 6", 80 times per minute, with a discharge varying from 6" to 8",

through a No. 9 slot-punched, Russian-iron screen, crushing 1¾ tons per stamp per twenty-four hours. The battery is supplied with an inside plate, about 6″ wide, attached to the screen; the latter is set slightly inclined. The screen-frame leaves about 4″ at the upper end of the mortar-front open, in front of which and reaching nearly to the lip is a canvas curtain. The apron-plate is 5′x4½′, set on a grade of 1″ in 11″; below the apron is a drop-box, from which the pulp passes to the sluice-plates; these are 10′ long by 15″ wide. The aprons are scraped every day with rubbber belting, and the plate on the screen is cleaned once or twice a week. In dressing the plates, brine with an addition of sulphuric acid is used. About 20% of the amalgam is saved in the batteries, and about 80% on the plates. Neither concentrators nor canvas-tables are used. One tank of quicksilver is used every six months, using twenty stamps.

No. 14. *Shasta County.*—The ores carries 1½% of iron and copper sulphurets, besides free gold, averaging $9 per ton. There are 30 stamps, weighing 850 lbs. each, supplied with Challenge feeders, working from the second stamp. These stamps are hung and dropped somewhat at variance with the usual custom, No. 1, the end stamp on the left, being placed 1″ farther from the side than is No. 5, the end stamp on the right, and the sequence of the drop is 5, 4, 3, 1, 2; the stamps never rising out of the water. It is claimed that by this arrangement a better swash is obtained in the battery. The stamps drop 5″, 92 times per minute, with a discharge of 6″ to 7″, and crushing 2 tons per stamp per twenty-four hours. The mortar is supplied with front and back inside plates. The apron-plate is 4′x4′, set on a grade of 1¼″ to the foot, followed by a double set of sluice-plates, 16″ wide and 16′ long, with a grade of 1″ to the foot. The apron-plate is kept rather wet with mercury by frequent dressing. Burr-slot screens, Nos. 40 and 45, are used. About 66% of the amalgam is derived from the battery. The pulp is concentrated on four Triumph and ten Frue vanners, and is then passed to two canvas-platforms, 36′ and 24′ long, respectively, and 20′ wide, divided into sections 2½′ in width, covered with twill instead of canvas, which is said to give equally good results, and is considerably cheaper. These tables have a grade of 1¼″ to the foot. The plates are scraped once a day, and the mill is cleaned up twice a month. The company chlorinate their own sulphurets, roasting in a small two-hearth furnace, with a capacity of one ton per twenty-four hours.

No. 15. *Sierra County.*—On account of topography, the ore has to be elevated by a lift to the top of the mill. The stamps weigh 850 lbs. each, and drop 5″, 80 times per minute, with a 6″ discharge through No. 7 slot-cut screens. The cams, bosses, and tappets are steel; the shoes and dies iron. The apron is 4′x4′, with a grade of 1¾″ to the foot, and is followed by a double sluice-plate, 16″ wide, 12′ long, and pitched 1½″ to the foot. The plates are not scraped at regular intervals; in dressing them, lye is used occasionally. The plate on the screen, 6″x52″, is cleaned every other day. About 86% of the amalgam is saved in the battery, and the tailings only show a trace of gold. Johnston concentrators receive the pulp from the plates; these concentrators are run with 110 to 112 side-strokes per minute, the belt revolving once in seven minutes. The waste from the concentrators is forced to a

higher level by an "ejector," and then passes through a pointed box. The heavy material is then passed through a series of drop-boxes and discharged into the river.

No. 16. *Sierra County.*—The ore carries a considerable amount of clay, and is delivered to the mill over an incline track to a Blake crusher. The stamps weigh 850 lbs. each, and make 75 to 78 drops of 6″ per minute, with a discharge varying from 7″ to 9″, using a No. 10 slot-cut screen. The inside of the mortar is furnished with front and back plates, the former 8″, the latter 4″ wide. Cast-iron shoes and dies are used, crushing 1½ tons to the stamp per day. The order in which the stamps drop is 1, 5, 2, 4, 3. The apron-plate is the width of the mortar, is 6′ long, and is set to a grade of 2½″ to the foot, followed by 12′ of sluice-plates, 14″ wide. As there are but few sulphurets, no concentrators are in the mill. The apron and sluices are dressed every day, but only scraped once a month; cyanide of potassium is used in dressing the plates. The battery is cleaned once a month. About 5 lbs. of quicksilver is used to every 1,500 tons. About 70% of the amalgam is obtained from the battery.

No. 17. *Tuolumne County.*—This mill of 10 stamps crushes quartz containing little or no free gold, but with 3 per cent of sulphurets, chiefly iron. The stamps weigh 1,000 lbs. each (fed by self-feeders), working at 96 drops of 6″, crushing 2½ tons per stamp. Chrome steel shoes and dies are used, which wear about 1″ per week. No. 30 brass wire screens are used, the screen having a slight inclination, 10°. There are no plates used on the inside of the battery, and only one apron-plate, 4½′x 6′, to each five stamps, which is dressed daily. It is set to a grade of 1¾″ to the foot. Nearly all of the amalgam is derived from this apron. The pulp passes from the aprons through a series of troughs to four Frue concentrators with corrugated belts, using a large amount of water. These are succeeded by wooden troughs, 8″ wide at top, spreading to 16″, which divide into three troughs, carrying equal amounts of pulp, thinned by adding 3 miner's inches of clear water above the forks. These deliver into a V-trough running at the head of a canvas-platform, divided into twelve sections, 22″ wide and 75′ long, set on a grade of 1½″ to the foot. These tables are covered with No. 7 duck, which lasts 90 days. The V-trough has extending over its entire length a square 8″ trough, for clear water. The pulp flows from the V-trough through 1″ auger-holes (two to each section of the canvas), supplied with wooden plugs to regulate the flow of the pulp, the clear-water trough being similarly supplied. Every half hour the flow of the pulp is arrested on a section, while the flow of clear water is continued until the lighter sands are washed off, leaving the sulphurets on the canvas. The flow from the tables is delivered into V-boxes running across the end of the canvas-platform, with a grade from the outer edges to the center, and delivered to a second canvas-table trough, where it undergoes a similar treatment. After removing the lighter sands from the upper platform, a sheet-iron pan is placed below the end, which extends to a separate trough, into which the sulphurets adhering to the canvas are swept with the aid of the flowing clear water, and conveyed to a settling-tank divided into two sections in such a way that when one section is filled, the sulphurets are run into the second section, allowing the first to be

shoveled out, each section being treated consecutively. When No. 1 is being swept, No. 2 has the pulp-flow shut off, making the operation continuous. After sweeping down a section, the plugs from the pulp trough are removed and the clear water shut off, permitting the concentration to be renewed. It requires the attention of two persons, night and day, to attend to these two platforms. Everything but the sweepings pass over the second canvas-platform, and then go to waste. The sweepings from the second table are treated in a manner similar to those

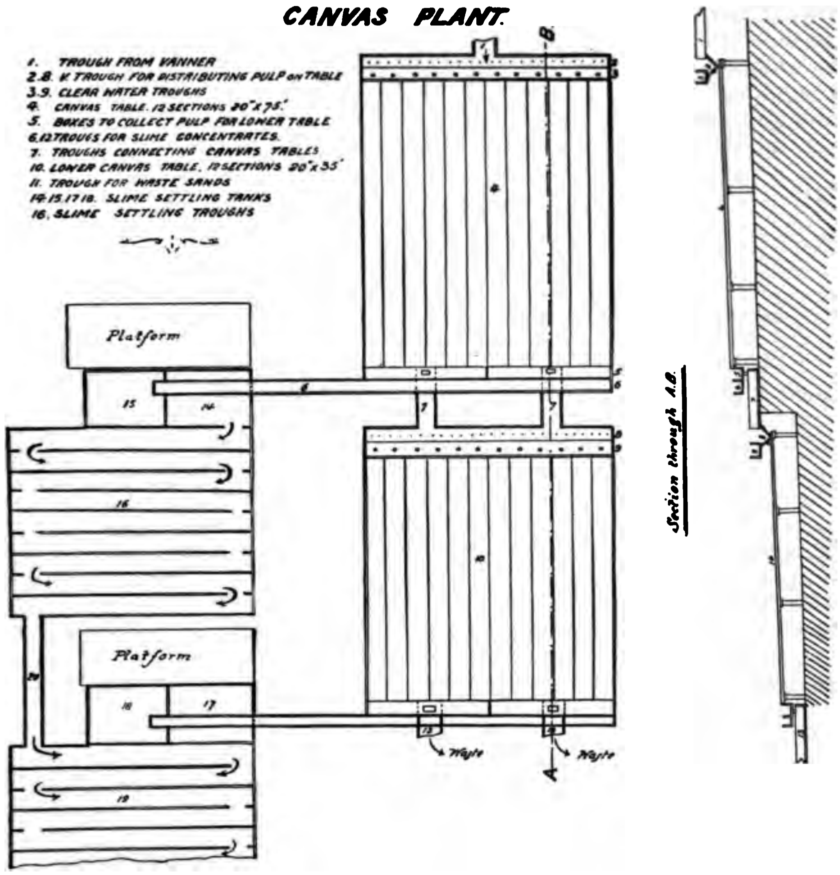


FIG. 50.

from the upper one. Accessory to these two sulphuret settling-tanks are two large slime settling-tanks; these are divided into twenty sections, 2′ square, 20′ long, divided from each other by 2″ plank extending entirely across the tank. Within 2′ of the ends, on alternate sides of these divisions, slots 4″ deep and 2′ wide are cut to permit the water, which is clouded with the fine slimes, to pass from one division to the other. At the end farthest from the entrance, the water (still somewhat clouded) runs to waste. The slime water from the first settling-tank passes through the second. The sulphuret settling-tanks are shoveled

out every few days, while the slime settling-tanks remain undisturbed for months. The material from the vanners, upper and lower settling-tanks, and sometimes from the slimes, is mixed by weight before going to the chlorination works. There are 2,400 sq. ft. of canvas-tables and 5,000 sq. ft. of settling-floor.

No. 18. *Tuolumne County.*—This mill is working on quartz carrying both free gold and sulphurets. The ore is delivered from the mine at the top of the mill, into a general ore-bin, for the entire 40 stamps, after passing through the rock-breaker. The bin has a capacity of 650 tons, and delivers the ore into the self-feeders direct. The stamps weigh 850 lbs. each, and steel shoes and dies are used. Each stamp has its separate guide, made of two blocks of hard maple, fitted together and bored through to receive the stem. The front block is first put in place, the stem set in, and the rear block dropped in behind a cast-iron piece,

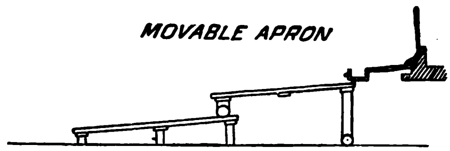


FIG. 51.

which is secured by wooden wedges driven in from above, so that when required it can easily be removed. The pulp from the battery falls over a 9″ silvered plate, the width of the mortar, into a box 12″ square, supplied with six 1″ holes, 8″ apart, near the front lower edge, that permit the pulp to flow onto a 4′x 5′ silvered plate, divided in two parts by a wooden strip down the center. The fall from the box to the plate is 3½″. The apron-plate is mounted on a carriage, which can be pushed back, giving access to the battery, the 4″ grooved wheels in front running on a half-round iron strip placed on the sides of the lower plate-frame. From the movable apron the pulp passes over 12′ of plates, divided into three 4′ sections, with a dividing strip down the center. Sixteen concentrators are used.

No. 19. *Tuolumne County.*—This mill presents some peculiarities in its construction. There are 10 stamps of 850 lbs. each, with steel shoes and dies. The stamps are given 100 drops per minute, dropping 4½″, with only 2½″ discharge to commence on, through a No. 50 brass-wire screen. The rock-breaker (Wheeler pattern) and ore-bin are set on a rock foundation, the frames being entirely disconnected from the rest of the building, to counteract the vibratory motion of the crusher. In placing the mortar-block and mortar, the space around the block was filled in with concrete, and a double thickness of tanned belting laid between the block and the mortar, after which a fire was built in the latter until it settled into the belting. The wear of the shoes and dies is about 1″ per month, and the duty of the stamps 1⅕ tons per stamp. When the dies are partially worn, a 2″ iron plate is placed under them, to maintain a regular discharge. The inside of each mortar is provided with cast-iron side plates and a sheet-iron covered board at the back, to prevent wear on the mortar. The apron-plates are set 12″ out from the mortar. The pulp from the battery passes over a 9″ plate into double-

pointed boxes of iron, bolted on the front of the mortar, and thence through a couple of 2″ pipes to a spreader and to the silver-plated apron. The apron is followed by double sluice-plates, each 2′ wide and 10′ long, all set on a grade of 1¼″ to the foot. Two thirds of the amalgam is obtained from the battery. No concentrators are used.

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## SPECIFICATIONS FOR A FORTY-STAMP GOLD MILL (WATER POWER).*

### MACHINERY.

*Water Wheels and Pulleys.*—One water wheel, 6′ in diameter, to drive the battery; the wheel to be supplied with a shaft, boxes, collars, gate, and nozzle, automatic governor, and a pulley 36″ in diameter, grooved for 1½″ manilla ropes.

One driving pulley, 12′ in diameter.

One idler pulley, 48″ inches in diameter, grooved for one 1½″ rope, and fitted with shaft and boxes.

One slack-tightener pulley, 48″ in diameter, grooved for one 1½″ rope, and fitted with shaft, boxes, carriage, track, and counterbalance weight.

The rope for transmission is to be put on in one piece, passing around the idler and slack-tightener (which are to be set on an angle in such a way that they will take the rope from one side of one of the main pulleys and pass it on to the opposite side of the other pulley), thereby making but one splice in the whole rope, which will be kept in constant tension by the slack-tightener.

One wheel, 4′ in diameter, to drive the rock-breakers; the wheel to be supplied with a shaft, boxes, collars, gate, and nozzle, and a pulley 34″ in diameter, grooved for one 1½″ manilla rope.

One driving pulley, 60″ in diameter.

One idler pulley, 30″ in diameter, grooved for one 1½″ rope, and fitted with shaft and boxes.

One slack-tightener pulley, 30″ in diameter, grooved for one 1½″ rope, and fitted with shaft, boxes, carriage, track, and counterbalance weight; rope to be put on similar to that for the battery.

One wheel, 36″ in diameter, to drive the concentrators; the wheel to be supplied with shaft, boxes, collars, gate, and nozzle, automatic governor, and a pulley 16″ in diameter, grooved for one 1″ manilla rope.

One driving pulley, 48″ in diameter.

*Forty-Stamp Battery;* stamps to weigh 850 to 900 lbs. each, arranged to run in eight batteries of five stamps each, by belts and friction clutch pulleys from battery line shaft.

Eight high cast-iron *Mortars*, single discharge, each to weigh about 5,000 lbs.; to be planed all over the bottom, and faced where the apron joins on; eight holes to be accurately cored in the base for 1½″ anchor bolts. Each mortar to have five cast-iron linings. The aggregate weight of these linings is about 500 lbs. per mortar.

Eight cast-iron *Aprons*, to be faced where they join on to the mortars, and fastened in place with ¾″ bolts.

* From the VIIIth Report of State Mineralogist, 1888, p. 728.

Eight sugar pine *Screen Frames*, to have iron facings put on the ends where the keys bear against them; the edges to be fitted with dowel-pins to join them to the inside plate-block.

Sixteen inside *Plate-Blocks*, two sets, one to be 6″ high, and the other to be 4″ high; to be well fitted into the mortars, and to have plates fitted and fastened on with brass screws; blocks to be bolted together to keep them from splitting, and to be fitted with iron facings where the keys bear against them, and well fitted to the screw frames.

Eight brass wire *Screens*, No. 30 mesh, to be fastened on to the screw frames with copper tacks.

Sixteen gilt-headed *End Keys*, for screen frames, to be well fitted in place.

Sixteen *Bottom Keys*, for screen frames, to be well fitted in place.

Sixty-four *Foundation Bolts*, for mortars, to be $1\frac{1}{2}$″ in diameter by 30″ long, with hexagon nuts on the top ends and steel keys in the bottom ends.

Sixty-four wrought-iron *Washers*, 4″x4″x$\frac{3}{4}$″, for bottom ends of foundation bolts.

Eight sheets of *Rubber*, $\frac{1}{4}$″ thick by 30″ by 60″, for mortar foundation. Mill blankets tarred may be used in place of rubber.

Forty chrome steel or cast-iron *Dies*, 9″ in diameter by 7″ high, with square base well fitted into the mortars, 10″ from center to center.

Forty chrome steel *Shoes*, 9″ in diameter by 8″ high, with tapered shank $3\frac{3}{4}$″ in diameter at top end, $4\frac{3}{4}$″ in diameter at bottom end, by 5″ long, to fit into the stamp-heads by being covered with dry, hard pine $\frac{3}{8}$″ thick; this being driven in by being allowed to drop a few times on the bare die.

Forty chrome steel *Stamp-Heads*, 9″ in diameter by 17″ long, with a conical socket cored into the lower end, 4″ in diameter at inner end and $5\frac{1}{8}$″ in diameter at the outer end, and $5\frac{1}{2}$″ deep, and a conical socket cored and actually bored out to fit the tapered end of the stamp stem, $2\frac{7}{8}$″ in diameter at inner end, and $3\frac{1}{4}$″ gauge at the outer end, by 6″ deep. Transverse rectangular keyways are to be cored through the stem-head, 1″x$2\frac{1}{2}$″, for loosening the shoes and stems.

Two steel *Loosening Keys*, $\frac{7}{8}$″ thick by 1″ at the point (2″ at the head) by 18″ long, for loosening the shoes and stems.

Forty best refined iron or mild steel *Stems*, turned perfectly true, full length, $3\frac{1}{4}$″ gauge by 14′ long, to be tapered on both ends to accurately fit the stamp-heads. Each stem weighs about 360 lbs.

Forty chrome steel, double-faced *Tappets*, 9″ in diameter by 11″ long, with a steel gib and two steel keys accurately fitted in place; both faces to be turned true; tappets to be bored with the gibs in place to accurately fit the stems, and to be counter-bored opposite the gibs by moving the center $\frac{1}{4}$″ away and, with diameter $\frac{1}{8}$″ less than the bore, taking a cut $\frac{1}{8}$″ deep. Each tappet weighs 112 lbs.

Eight *Upper* and eight *Lower Guides*, with cast-iron frames; guide-blocks to be made of good, dry maple timber and well fitted in place; the guides may also be made entirely of wood.

Four extra quality, mild steel *Cam-Shafts*, turned true full length, 5½" gauge diameter by 14' long; key-seated for cams and pulley; key-seats must not run through the bearings.

Ten heavy *Corner Boxes*, 5½" gauge bore; eight of them to be 12" long, and two to be 20" long; all of them to be planed all over the bottoms and backs, and furnished with bolts 1" in diameter, to fasten them to the battery frame.

Forty double-armed, chrome steel *Cams*—twenty right and twenty left hand—to be made 29" long over all, the hub to be 11" in diameter and 5½" through the bore; the lifting faces to be 2½" wide, and ground smooth; the hubs to be bored to fit the shaft accurately, and properly key-seated and fitted with steel keys, and each marked to their respective places, giving them a combination as follows: Counting from the left-hand side. when facing the battery, throughout the full ten stamps of each cam-shaft, No. 1 cam will drop its stamp first; then Nos. 8, 4, 10, 2, 7, 5, 9, 3, and 6 consecutively. This is the order: 1, 4, 2, 5, 3. Each cam weighs about 158 lbs. The curve of the face of the cam is the involute of a circle, usually slightly modified.

Four pairs of cast-iron double *Sleeve Flanges*, for wood pulleys; flanges to be 36" in diameter, and 14" through the bore; to be turned all over the inside, where they fit on the wood; the outside flange is to be bored and fitted to the sleeve and fastened with a gib-headed steel key; the hub to be bored and fitted to the cam-shaft and fastened with a steel key.

Four *Wood Pulleys*, 72" in diameter by 17" face; to be made of best kiln-dried sugar pine, and all joints to be filled with white lead in oil; the cast-iron flanges to be well fitted on and bolted with twelve ⅞" bolts.

Eight wrought-iron *Collars*, for cam-shaft, 5½" bore, fitted with two steel set-screws in each.

Eight wrought-iron *Jack-Shafts*, 3" in diameter by 60" long; black finish.

Sixteen cast-iron *Jack-Shaft Side Brackets*, with four lag-screws, ⅝" by 6", for each, to fasten them in place.

Forty open *Latch Sockets*, lined with leather.

Forty wood *Finger-bars*, to be fitted and bolted to the above sockets, and furnished with wrought-iron caps and handles.

A complete set of *Water-Pipes* for a battery of forty stamps, with all fittings, cocks, and connections.

*Bolts and Washers for Battery Frame.*—Six brace rods, 1¼"x25', 7" between two nuts; 6 brace rods, 1¼"x12', 6" between two nuts; 26 bolts for mudsills, 1"x30"; 24 bolts for yokes, 1"x28"; 24 bolts for yokes, 1"x52"; 48 bolts for guide girts, 1"x32"; 4 bolts for knee beam, 1"x28"; 36 splice bolts for mudsills, ⅞", 16" between head and nut; 12 splice bolts for tail girt, ⅝", 9½" between head and nut; 32 bolts for mortar-blocks, 1", 59" from point to point; 64 bolts for mortar-blocks, 1", 65" from point to point; 24 joint bolts for posts, 1", 35" between two nuts; 6 joint bolts for knee posts, 1", 45" between two nuts; 6 joint bolts for knee posts, 1", 35" between two nuts; 24 joint bolts for knee beams, 1",

43″ between two nuts; 10 joint bolts for tail girts, 1″, 21″ between two nuts; 24 cast-iron washers for 1¼″ rods; 514 cast-iron washers for 1″ bolts; 72 cast-iron washers for ⅞″ bolts; 24 cast-iron washers for ⅝″ bolts; 40 sheet-iron washers, 3½″ square by ¼″ thick, for 1″ joint bolts.

*Battery Line Shafting and Pulleys.*—One shaft, 5½″ gauge by 18′ long, properly key-seated; one shaft, 5″ gauge by 15′ 6″ long, properly key-seated; one shaft, 5″ gauge by 17′ long, properly key-seated; one shaft, 4″ gauge by 17′ long, properly key-seated; two shafts, 3″ gauge by 10′ 6″ long, properly key-seated; two face couplings, 5″ gauge, properly fitted and keyed in place; one face coupling, 4″ gauge, properly fitted and keyed in place; two face couplings, 3″ gauge, properly fitted and keyed in place; two babbitted boxes, 5½″ gauge; three babbitted boxes, 5″ gauge; two babbitted boxes, 4″ gauge; two babbitted boxes, 3″ gauge; all of the above boxes to be made of the same height, planed all over the bottoms, with drip cups cast on to the sides, and furnished with suitable bolts to fasten them to the 16″ battery knee beams; two collars for 5½″ shafting, with two steel set-screws in each; four friction clutch pulleys, 48″ in diameter and 16½″ face, complete, with levers and connections; pulleys to be fitted to line shaft in their proper places, with phosphor-bronze bushings, the drivers to be properly keyed on with steel keys; one pulley, 6′ in diameter, grooved for three 1½″ manilla ropes, pulley to be well balanced and keyed to the shaft with a steel key.

*Water-Pipes.*—Sufficient 3″ pipes and fittings to connect battery pipes with feed-water tanks.

*Traveling Hoist.*—One traveling crab, with track-iron and rails, to extend full length of battery.

One 2-ton Weston's differential chain-block.

*Ore-Feeders.*—Eight Challenge self-feeders, complete, for batteries, with all connections.

*Ore-Bin Gates.*—Eight ore-bin gates, 18″x24″, for fine ore, with guides, racks, pinions, shafts, boxes, hand-wheels, and bolts.

Three ore-bin gates, 24″x36″, for coarse ore, with guides, racks, pinions, shafts, boxes, hand-wheels, and bolts.

*Sluices and Aprons.*—Eight cast-iron aprons, 54″ wide by 56″ long, to be fitted under the lip of the mortar apron.

Eight silver-plated copper plates, 54″x56″x⅛″, to be made of best Lake Superior copper, and to have one ounce of silver per square foot; plates to be fitted into the cast-iron aprons, and fastened by strips of wood on the sides, which are bolted to the sides of the apron.

Eight cast-iron sluices, 54″ wide by 12′ long, to be made into two sections and bolted together by flanges, the lower section to have a quicksilver trap or trough cast on to the end, extending the full width of the sluice, and to have a connection made for a 2″ pipe to conduct the pulp to the dividing tanks, and thence to the concentrators.

Twenty-four silver-plated copper plates, 54″x48″x⅛″, to be made of best Lake Superior copper, and to have one ounce of silver per square foot; plates to be fitted into the sluices, overlapping at the joints, and to be fastened in place in the same manner as those in the apron.

There are to be eight silver-plated copper shaking-tables, one for each battery, placed below the apron-plates. These tables consist of a light

iron framework suspended upon movable springs. This table is given a longitudinal oscillation by means of eccentrics.

*Dividing Tanks and Pulp Pipes.*—Eight cast-iron dividing tanks, 10" long by 8" wide by 6" deep, with 2" pipe connection in one end and two 1½" pipe connections in the other end, each to have a wooden swinging tongue put in so as to direct the pulp to either of the 1½" pipes, or a part to the one and a part to the other. The tanks are to be connected with the sluices by 2" pipes, and with the concentrators by 1½" pipes.

*Inside Plates and Blocks.*—Three wooden blocks for each mortar, to be 3", 4½", and 6" high, respectively, to be fitted into the mortars under the screw frames; each block to have iron facings, fitted in flush and screwed on where the keys come, and to have a silver-plated copper plate bent to the proper shape and screwed on with silver-plated brass screws; the copper plates to be made of best Lake Superior copper, 6"x 50"x $\frac{3}{16}$", and to have one ounce of silver per square foot.

*Concentrators and Shafting.*—Sixteen endless belt concentrators, complete, with water-pipes and fittings to connect with supply tanks. All sulphuret tanks, complete, to be made of good redwood lumber.

One piece of shafting, 2½"x 16'; six pieces of shafting, 2"x 16'; three pieces of shafting, 2"x 10'; eight face couplings, 2"; four babbitted boxes, 2½", with bolts for 8' timber; eighteen babbitted boxes, 2", with bolts for 8" timber; two collars, 2½", with steel set-screws; two collars, 2", with steel set-screws; one pulley, 48" in diameter, grooved for one 1" rope, and properly fitted and keyed with a steel key to 2½" shaft; two pulleys, 6" face by 36" in diameter, properly fitted and keyed with steel keys to the 2" shaft; sixteen pulleys, 4" face and 10" in diameter, properly fitted and keyed with steel keys to the 2" shaft; sixteen loose pulleys, 4" face by 10" in diameter, properly fitted to the 2" shaft; sixteen collars, with steel set-screws for same.

*Rock-Breakers and Shafting.*—Two rock-breakers, 9"x 15"; one piece shafting, 4"x 16'; one piece shafting, 3½"x 16'; one piece shafting, 3"x 16'; one face coupling, 3½"; one face coupling, 3"; three babbitted boxes, 4", with bolts for 10" timber; two babbitted boxes, 3½", with bolts for 10" timber; two babbitted boxes, 3", with bolts for 10" timber; two collars, 4", with steel set-screws; one pulley, 48" in diameter, grooved for 1" and 1½" manilla rope, and properly fitted and keyed to the 4" shaft, with a steel key; three pulleys, 20" straight face by 20" in diameter, properly fitted and keyed to the shafting.

*Clean-up Barrel.*—One clean-up barrel, 24" inside diameter by 48" inside length, to be made of cast-iron 1½" thick, with two discharge openings, 5½" in diameter, in the sides diametrically opposite each other, the heads and discharge doors to be accurately fitted; journals to be 4" gauge, cast on to the heads; one tight and one loose pulley, 7" face by 30" in diameter; two babbitted boxes, 4" gauge; one driving pulley, 6" in diameter by 14" face.

*Batea.*—One batea, 48" in diameter, with gears and hangers complete, and tight and loose pulleys, 4½" face by 16" in diameter; one driving pulley, 9" face by 21" in diameter.

*Machinery for Clean-up Room.*—One clean-up pan, 24" inside diameter, with tight and loose pulleys.

One driving pulley, 8″ face by 16″ in diameter.

One cast-iron washing-tank, 24″ by 30″ by 24″ deep, with three pipe connections for drawing off water.

One cast-iron washing-tank, 30″ by 36″ by 24″ deep, with three pipe connections for drawing off water.

One cast-iron washing-tank, 30″ by 54″ by 30″ deep, with three pipe connections for drawing off water.

One marble top, complete, for washing-tanks.

One side washstand, with pipes and fittings.

All pipes and fittings necessary to bring water to the clean-up pan and washing-tanks.

*Retort and Melting Furnace.*—One retort, 10″x36″, inside dimensions, with amalgam trays, condenser, catch tank, furnace front, bearers, bars, smokestack, and base plate, guy rods, dampers, binders, and all pipes and fittings to bring water to the condenser.

One cast-iron melting furnace, complete, with doors, grate-bars, bearers, cast-iron shell, and damper.

Two bullion molds for 500 and 750 ounces.

Four black-lead crucibles, No. 16, with covers.

One crucible tongs for No. 16 crucible.

One skimmer for bullion.

*Transmission Ropes and Belts.*—Six hundred feet best manilla or cotton rope, 1½″ in diameter, to drive battery line shaft.

Two hundred and fifty feet best manilla rope, 1½″ in diameter, to drive rock-breaker line shaft.

One hundred and fifty feet best manilla rope, 1″ in diameter, to drive concentrator line shaft.

Two hundred feet best rubber belting, 16″ by 5-ply, for batteries.

One hundred and eighty feet best rubber belting, 10″ by 4-ply, for rock-breakers.

Thirty-two feet best rubber belting, 7″ by 4-ply, for clean-up barrel.

Sixty-five feet best rubber belting, 6″ by 4-ply, for batea.

Thirty feet best rubber belting, 6″ by 4-ply, for concentrator shafting.

Four hundred and twenty feet best rubber belting, 3″ by 4-ply, for concentrators.

Thirty feet best rubber belting, 3″ by 4-ply, for clean-up pan.

## BUILDINGS, AND ERECTION OF MILL, ETC.

*Stonework.*—All foundations and retaining walls to be built of large stone, properly banded, and well laid in cement mortar, composed of ten parts good, clear sand, two parts good quality of lime, and one part best Portland cement, special care being taken to keep all dirt or clayey material excluded; all exposed faces of retaining walls to be well pointed up and finished with the same material.

*Ore-Bins.*—Mudsills to be made of 12″x14″ timbers, laid flatwise; foundation posts to be made of 14″x14″ timbers; sills, posts, and caps for ore-bins proper to be made of 12″x12″ timbers, the posts to be boxed 1″ into the sills and caps; braces for incline bottom, to be made of 10″x12″ timbers; supporting braces to be made of 8″x12″ timbers. All planking to be 3″ thick and lined throughout with 1″ boards, to break joints over the planks.

*Battery Frame.*—Mudsills to be made of 14"x16" sugar pine, or good yellow pine free from sap; to be well bedded in concrete, which must be put on the clean bedrock. Linesills to be made of 12"x16" and 20"x16" sugar pine or yellow pine, of good quality, to be well bolted down to the mudsills.

Mortar-blocks to be made of two pieces each, to be 30" thick and wide enough to fill space between the linesills and battery posts; all to be sized and well fitted. The timbers for mortar-blocks are to be accurately fitted together and secured with six 1" bolts, and two oak keys, 4" wide by 5" thick at the point and 6" at the head. Keys to be accurately fitted and firmly driven. Blocks to be sized and finished above the floors.

Yokes to be made of 10"x10" timber, well fitted and bolted to the linesills and battery posts.

Battery posts to be made of 12"x24" and 20"x24", good quality pine timber, to be dressed all over, and bolted down to the linesills with 1" joint bolts, the large posts to be made with double tennon on the bottom. The knee beams to be made of 12"x16" timber, dressed all over. The knee posts to be made of 12"x16" timber, dressed all over. The stringer on top of the knee posts to be made of 12"x16" timber in two pieces, to be spliced with a ship splice 3' long, stringer to be dressed all over. Knee posts to be framed into stringer with double tennons; outside stringer at end of knee beams to be made of 8"x12" timber in two pieces, spliced with ship splices in center 3' long, and to be dressed all over.

Bottom guide girt to be made of 12"x16" timber, dressed all over, one piece for each twenty-stamp battery, and to extend past the outside posts 12"; the top girt to be made of 12"x14" timber, dressed all over, and made the same length as the lower ones; all braces to be made of 8"x12" timber, dressed all over, and framed with double tennons; no keys are to be used in braces or guide girts, but they must be accurately fitted without.

All boxing about battery frame to be ½" deep, and where braces or knee beams are smaller than the timbers they frame into, they must be housed in ½" deep; *i. e.*, the timber must not be boxed out clear across.

The cam-shaft is to be set 4¾" from the center to the center of the stems.

A 2" plank floor is to be put on top of the knee beams, which is to be planed on the under side; also, a 2" double board floor to be put in back of the battery, on about the same level as the knee beams.

The whole battery frame to be painted with two coats of light-cream paint, properly mixed with oil, and the wood pulleys and guides to be painted blue, the iron work to be painted black. The out-board bearing frame to be made of 12"x16" timber, planed all over, well framed and bolted together, and anchored to a solid stone foundation, as shown in plan, and to be painted same as battery frame.

*Water Wheel Frames* are to be made of 12"x12" lumber throughout, well anchored down to a stone foundation. That part of the frame which comes above the floor is to be dressed and painted the same as the battery frame.

The water wheels are to be housed with tongued and grooved lumber, 4" wide.

### BUILDINGS.

*Frame Work.*—Ore-house main frame is to be made of 8"x8" timbers throughout, with 3"x6" girts and studding.

Battery and concentrator rooms frame is to be made of 8"x10" posts and chords, 6"x10" sills, 8"x8" principal rafters and straining beams, 4"x8" truss braces, and 3"x6" girts and studding.

Clean-up, sulphuret, and water wheel rooms main frames are to be made of 8"x8" timbers, with 3"x6" girts and studding.

*Floors.*—Ore-house floors to be made of one thickness of 2" planks.

Battery, concentrator, and water wheel rooms floors are to made of 1"x 8" lumber, double thickness, surfaced on top, to be supported on 3"x 6" joists 18" apart.

Sulphuret and clean-up rooms floors are to be made of concrete laid on top of a heavy wood floor, which is to be supported on foundations made of 8"x 8" timbers.

*Roofs.*—All roofs are to be made with 2"x 8" rafters 18" apart, with 1"x 6" board 4" apart, and covered with No. 26 standing seam, painted, iron roofing.

*Walls.*—All walls are to be covered with 1"x 10" rustic.

*Cornices.*—All cornices are to project 24", measured horizontally from the walls of building, with a 12" frieze and a 5" facia made of dressed lumber.

*Windows.*—All windows, except those for sulphuret room, are to be made of twelve lights of 10"x 16" glass, and frames made to suit of dressed lumber, with casing outside 5" wide.

Twelve windows are to be put in the ore-house, seven windows in the battery room, six windows in the clean-up room, twelve windows in the sulphuret room, and five windows in the water wheel room.

*Skylights.*—Six skylights, made of twelve lights of 10"x 20" glass, to be put into the roof of the concentrator room.

*Doors.*—All doors, both sliding and swinging, to be 3'x 7'x 1¾" thick, with panels.

Two sliding doors are to be put in the ore-house, and one outside swinging door in the battery room; one swinging door leading from the battery room to the clean-up room; two sliding doors leading from the concentrator room to the sulphuret room; two outside sliding doors for the sulphuret room, and one outside swinging door for the water wheel room.

All doors to be set in good substantial casings, outside cased with surfaced lumber, and furnished with all trimmings and locks.

*Stairs.*—There is to be a flight of stairs at each end of the mill, one flight leading from the battery room floor to the floors above, and one flight of stairs from the battery room floor to the concentrator room floor.

All stair stringers to be made of 2"x 12" lumber, and treads of 2"x 10" lumber.

*Hand Rails* are to be put on to the outside of all stairs and around the landings of same, also in front of the battery room floor and all other floors and platforms where there is danger of falling. All to be made of dressed lumber, well painted.

*Retort House and Assay Office*, to be 20′ wide by 48′ long, with a retort and melting furnace room, a weighing room, and a storeroom; the two latter to be lath and plaster finished, and the whole building to be finished similar to the mill buildings, with iron roof, rustic, etc.

*Paint and Whitewash.*—All buildings are to be painted on the outside with a good coat of brown mineral paint, and the window and door casings and cornices to be painted with two coats of white lead paint.

The mill to be whitewashed throughout the inside, including the building frame, ore-bins, etc.

*Tanks.*—There are to be two 4,000-gallon redwood tanks, 3″ stock, set up at the end of the mill upon strong timber foundations, and one tank 8′ wide by 10′ long by 4′ high, inside measurements, to be made of 3″ planks, with 8″ x 8″ frame; planking to be well fitted together, and properly caulked inside with oakum. The latter tank is to be set at the end of the last sluice-box coming out of the mill.

*Drain Boxes and Tailings Sluices.*—Battery sluices and aprons to be set on framework so arranged that the grade can be changed easily. This framework to be planed all over. Sluices and frames to be painted same as battery frame.

There will be a sluice in front of battery room floor, made of surfaced lumber; also to be painted and so arranged as to conduct any water away which drips from the floor.

There will be sluices put in under the concentrator room floor, two of which will be 6″ wide by 8″ deep, to run lengthwise to catch the tailings from the concentrators, and one to be 8″ wide by 10″ deep, to run crosswise and to take the tailings from the first two sluices, and conduct the same outside. All tailings sluices to have a fall of one in twelve, and to be made of 2″ lumber, well fitted and nailed together. Proper sluices from the clean-up room, to conduct water and tailings therefrom, must be connected to tailings sluices under concentrator room.

All sulphuret boxes, and drain boxes for concentrators, to be made of good quality of redwood lumber, 1¼″ thick, dressed on both sides, and well fitted and screwed together.

The weight of all parts is 240,000 lbs., and there are 325,000 feet of lumber in the building.

*Specifications for a canvas plant* are not considered necessary, as the construction is extremely simple and no standard has been adopted. Full descriptions are given in the preceding pages.

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